ROCK EXCAVATION

METHODS AND COST

BY

HALBERT POWERS GILLETTE

M. Am. Soc. C. E., M. Am. Inst. M. E.
E. M. School of Mines, Columbia University
Late Assistant New York State Engineer

NEW YORK
M. C. CLARK
13-21 Park Row
1904

Mining Budget
MCC

PREFACE.

The value of the mineral products of America annually exceeds $1,000,000,000. For explosives alone the sum of $20,000,000 is expended each year, and it is well within limits to say that fully twice as great a sum is paid out to drillers and blasters. When we consider the labor, the power and the powder required to mine and to quarry products in the aggregate so enormously valuable, we can not fail to be impressed with the scantiness of literature on the economics of rock excavation. A dozen years ago, when called upon to estimate the cost of some open cut rock excavation, I was astonished to find no text book that in the least served to guide me. I subsequently learned also that most of the matter to be found in the few books on blasting was either theoretic or too meagre to be of material value. What was true then has unfortunately remained true.

In the preparation of this volume, it was my original intention to give only condensed cost data on rock excavation—including in that term, quarrying, open cut work, trenching, tunneling, and underground excavation; but, as the writing progressed, it become more and more apparent that methods of doing work and "tricks of the trade" should occupy an important part of the book to make it at all satisfactory. I saw that my object should be not merely to furnish data whereby an inexperienced man might predict the cost of rock work with at least tolerable accuracy, but that it should be my aim also to indicate to the experienced manager how and where savings in the cost of production may be effected.

Taking my own notes and records of cost as a basis, I have added so extensively to them by correspondence that the reader will find not a little on costs that has never appeared in print before. In addition, I have abstracted scores of pages of facts and figures from the volumes of scientific and engineering periodicals, due acknowledgement of which appears throughout the text.

While the scope of the book is wider than at first sight appears desirable, since it includes quarrying, open cut excavation, trenching, subaqueous excavation, tunneling and underground excavation, still it should be remembered that the main elements of cost are of much the same *quality,* and that the differences are principally those of *quantity.* Thus, it may require eight feet of drill hole per cubic yard of tunnel excavation as compared with half a foot for open cut work. The cost of drilling per cubic yard is obviously much greater in the tunnel than in the open cut, but the methods of drilling and the cost per foot of drill hole may be practically identical. There is, indeed, much in common in all classes of rock excavation in spite of many detailed differences.

For these reasons, and because my training and experience have been both in mine and in quarry work, I have chosen for this book the broad scope indicated by its title. I am well aware of defects in the execution of my plan, but trust, however, that suggestions as to sources of information now unknown to me, and original data will be sent to me by readers who are interested in the progress of the art of economic excavation. There is probably not a man of any considerable experience who could not add his quota of valuable information, if he would; and, on the other hand, there is not one of us who "knows it all"—except the fellow who really knows very little. Men engaged in mining can learn from those whose excavation work is in the open air, and the reverse holds equally true. The mining engineer profits to-day by inventions developed under the direction of civil engineers, and thousands of civil engineers in turn are indebted to mining engineers. It is well for us not to forget that the first air drill, so useful in mining, was the development of work in a railway tunnel, and that the first railway was the development of work in a mine.

New York, Sept., 1904. HALBERT P. GILLETTE.

CONTENTS.

CHAPTER I.

ROCKS AND THEIR PROPERTIES.

Rock-forming Minerals.—All rocks are aggregates of one or more minerals or the disintegrated products of minerals. A mineral is an inorganic body having a definite chemical composition, as quartz, common salt, mica, and the like. There are about 1,000 distinct species of minerals, but fortunately the common rock-forming minerals number less than 30; and of these 30 perhaps 15 should be recognized at sight by everyone who aims to become an expert in rock excavation.

A small cabinet of the common species of minerals may be purchased for a few dollars from any of the dealers whose advertisements may be found in the mining and civil engineering journals. Such a cabinet when studied with the aid of a small book on mineralogy will enable one to undertake the work of rock excavation intelligently. It is often said that "a little knowledge is a dangerous thing"; but a little knowledge of fundamental principles, whether of geology or of any other of the physical sciences, is exactly the opposite of dangerous. A little knowledge of a few scientific facts, it is true, often leads to incorrect rules or generalization; but a knowledge of correct rules and fundamental principles, based upon the observation and study of many facts by experts, is the kind of little knowledge that no man can afford to be without. Thus, for example, geologists classify rocks according to their origin into two great classes, (1) sedimentary or stratified, and (2) crystalline, or igneous. A contractor, who had lost many thousand dollars on some rock excavation in the northern part of New York, once told me that he attributed his loss to a lack of knowledge of geology. His previous experience had been confined to the shales and sandstones of Pennsylvania. When he came to estimate the cost of excavating a granitic rock he made due allowance for its greater hardness and toughness as affecting the speed of drilling; but he failed to consider the fact that the absence of lines of stratification in the granite would necessitate placing drill holes much closer together than in shale or sandstone. The result was that not only the cost of drilling but the cost of explosives per cubic yard of granite excavation was practically double what he had counted upon.

Every mining man can cite many striking instances to show how ignorance of the elementary facts of geology and petrology have lead to serious underestimates of the cost of tunneling, shaft sinking and stoping. It is true that where an engineer, contractor or miner works all his life in one locality he becomes so expert in his knowledge of the methods and cost of rock excavation that he sees little practical value to himself in a knowledge of minerals, rocks or geologic principles. But when, possibly late in life, he goes to a new field of action, he is likely to lose his reputation, if not his money, through lack of a "little knowledge" of the fundamental principles of rock formation. The science which he has regarded as being too theoretical for him might have saved him had he possessed even a little of it.

I therefore repeat that the man who aims to become an expert in all kinds of rock excavation, should first learn to know the common species of minerals at sight, or by means of the simple tests described in books on mineralogy. It is beyond the province of this book, however, to describe rocks and rock forming minerals. There are several excellent books on that subject, with one or more of which the rock excavator should be familiar.

Hardness.—The resistance to scratching or cutting is termed hardness. Diamond is the hardest mineral known, as it will scratch all others. Talc is one of the softest minerals. Mineralogists use a scale of hardness as follows:

1. Talc.
2. Gypsum.
3. Calcite.
4. Fluorite.
5. Apatite.
6. Feldspar.
7. Quartz.
8. Topaz.
9. Corundum.
10. Diamond.

The sharpest point of a steel knife will not scratch quartz, but under considerable pressure will scratch feldspar. A very slight pressure on the knife will scratch talc or gypsum; indeed the finger nail will serve to scratch them. The test of hardness often serves to distinguish one mineral from another; for instance, iron pyrites and copper pyrites are similar in color, but while copper pyrites can be scratched with a knife, iron pyrites cannot. To distinguish sandstone from limestone it is often necessary merely to draw a sharp corner of the stone across a pane of glass. If a scratch is left in the glass the stone can not be pure limestone, since calcite, which has a hardness of 3, is the mineral forming limestone. Impure limestone may be a mixture of fine grains of quartz and calcite; but after a little experience in testing stones and minerals the eye will aid to such a degree in determining the species that the test of hardness becomes a very reliable one in many cases.

Hardness of course affects the speed of drilling in rock, although to a less degree than toughness. A tough rock is one that will stand a hard blow without splintering. Window glass is quite hard, almost as hard as tempered steel, but it is not very tough. Sandstone is hard, so far as its individual grains are concerned, but is often drilled with ease, since it usually lacks in toughness. Trap rock is both hard and tough, and makes drilling difficult, besides dulling the drill rapidly.

Rock Species.—Rocks may be classified as: (1) Sedimentary; (2) igneous, and (3) metamorphic. Sedimentary rocks have been deposited originally from suspension or solution in water; thus sand hardened into rock becomes sandstone; clay becomes shale or slate; gravel becomes conglomerate or pudding stone; wood fiber becomes peat or coal; shells of minute forms of sea life form limestone, or possibly lime in solution crystallizes out as does rock salt. The chief characteristic of sedimentary rocks is that they lie in beds or layers, one upon the other, often not cemented together in the slightest degree; and even where they appear to be solid and massive they can usually be split into slabs by wedging.

The igneous rocks have at one time been in a molten condition, and include the traps, porphyries, most granites, and all volcanic lavas, etc. Igneous rocks are often exceedingly tough, hard to drill, and are apt to break out in very irregular masses, sometimes of huge size and in other cases of too small a size to make cut stone masonry.

The metamorphic rocks, may be said to be a "cross between" the sedimentary and the igneous rocks, for they have been formed by chemical and physical changes in sedimentary rock under the influence of heat, water and pressure. For example, a dike of molten rock rising through a fissure in shale heats the surrounding shale to such a degree that, in the presence of confined water, the shale dissolves or melts, and, when it solidifies by subsequent cooling, a gneiss is produced. Some granites are known to have been made in this way by what is called metamorphic action. Marble is metamorphosed limestone, and quartzite is metamorphosed sandstone.

Joints.—All rocks are more or less split up along vertical or nearly vertical planes termed, joints, which greatly assist in the quarrying. Joints are the results of stress, due to shrinkage of the earth's crust upon cooling. This shrinkage has produced great compression in some places and tension in others, resulting in cracking of the rock masses at more or less regular intervals.

In limestone and in close grained shales the joints are often regular but so close as to be invisible until revealed by fracture or by weathering. In coarse grained sedimentary rocks, the joints are apt to be more open and irregular, running into one another or branching.

In sedimentary rocks there are generally two sets of joints running approximately at right angles, known as the dip-joints and the strike-joints. The "dip" of the rock is the angle that its bedding planes make with a horizontal plane. The "strike" is the line of intersection between an inclined plane and a horizontal plane; thus, if a sheet of cardboard be held at an incline in a basin of water, the line of the water surface along the face of the cardboard is the "strike." Since a quarry is usually worked to the dip of the rock, the strike joints, or "backs," form clean cut faces in front of the workmen as they advance; while the dip joints, or "cutters" form the side faces of the benches in the quarry. Thus it happens that in some sedimentary stone quarries, nature has provided blocks of stone, practically loose on all six faces; but, as a rule, the joints are so irregularly spaced as to require much plug and feathering work.

Igneous rocks, while not possessing bedding planes, also have nearly vertical joints, cutting at about right angles in most cases; but these joints are seldom so regular in spacing as in sedimentary rocks. In certain trap rocks the joints cause the rock to break out in vertical columns, often of great regularity; and in some cases the joints are so close together that upon firing the blast the rock comes down in chunks not much larger than a man's head, even where very little explosive is used; but, on the other hand, certain traps break up in very large chunks on blasting.

Veins and Beds.—Iron ores and coal occur for the most part in beds that originally were nearly horizontal, since they are of sedimentary origin. They are, or at one time have been, overlaid by sedimentary rocks.

Veins carrying the ores of the valuable metals are in some cases fissures or cavities that have been filled by hot waters carrying minerals in solution. These fissure veins exist in both sedimentary and in igneous rocks, but as a rule near dikes or sheets of igneous rocks that were at one time in a molten state.

Limestone.—As its name implies, limestone is the rock from which lime is made. It is seldom pure (calcite), but usually contains more or less clay and often silica. By dissolving some powdered limestone in nitric acid the amount of impurity may be roughly ascertained, for neither clay nor quartz is dissolved by the acid. In some cases it will be found that there is less limestone than clay in the so-called limestone. Due to the presence of impurities, few rocks vary more in compactness and appearance than does limestone. When pure, it is hard and crystalline; but friable, chalky deposits of limestone are not uncommon. The common color is a gray, or a blue gray, passing into white. Marble is a pure metamorphosed limestone. Dolomite is a magnesian limestone in which part of the lime has been replaced by magnesia.

Sandstone.—As its name implies, sandstone is sand cemented together. The cementing material is commonly silica or iron oxide (iron rust). When cemented by silica the rock is apt to be very tough and far more durable than when cemented by iron rust; and is consequently more difficult to drill. Sandstone generally contains enough iron oxide to give it a red or brown color. White sandstone, as well as dark blue, is quite common. Sandstone often contains enough clay to make it difficult to classify it; in such cases it is usually very fine grained and may be mistaken for slate, especially when it splits into thin slabs.

Shale.—Shale is really a baked clay or mud, generally yellow, brown or black in color, and easily split into leaves. Under great pressure shale has often been converted into slate in which the pressure has forced the long particles into a position perpendicular to the line of pressure, and imparted lines of cleavage entirely independent of the original bedding planes of the shale. Shales often contain silica and lime to a degree that makes their classification puzzling. Dry clay upon absorbing water swells, so that clays which have been only partly baked into shale absorb water upon exposure and go to pieces; moreover, the swelling often causes great difficulty in tunnelling or shaft sinking, since the force developed by absorbing water may crush or displace the timbers used in lining.

On the other hand well hardened shales do not swell upon exposure and, if not exposed to great changes in temperature, have a long life. It often becomes a very important economic question to decide whether a tunnel through shale should be lined with concrete or masonry, and many civil engineers have made serious blunders through lack of scientific knowledge of the properties of the different shales. If an engineer has had experience only with half-baked shales, he is apt to line with masonry every permanent tunnel that he builds in shale, regardless of the kind of shale; while, on the other hand, an engineer whose experience has been with well baked shales may err by failing to provide lining for tunnels in half-baked shales. I could mention several instances of economic errors of this kind, but I have perhaps emphasized sufficiently the advantage possessed by the engineer who has a good working knowledge of rocks and of geology.

Granite.—Granite is composed of quartz, feldspar and mica; the quartz acting as the cement binding the whole together in a crystalline mass. When the percentage of feldspar is large the granite is said to be porphyritic. The mean specific gravity is about 2.65.

Porphyry.—Porphyry is an overworked name applied by miners to almost any igneous rock whose real name they do not know. Quartz-porphyry, or felsite, is composed of quartz and feldspar; when the quartz is visible as well-marked grains or crystals, the rock is generally called quartz-porphyry; but when the quartz and feldspar are so intimately mixed as not to be readily distinguished, the term felsite is more often used.

Trap.—Trap is another overworked name, that is commonly applied to fine grained rocks of igneous origin. Among the trap rocks are: Diabase, diorite, basalt, etc. The trap rocks usually have irregular joints, and, while on account of their toughness they may be excellent material for macadam, they are seldom fit for building purposes, except when crushed and used in concrete. The Hudson River trap, diabase, has a specific gravity of about 2.95.

Weight and Voids.—Civil engineers commonly measure rock excavation by the cubic yard in place before loosening, whereas mining engineers generally use the ton of 2,000 pounds, as the unit of rock and ore measurement. In view of this fact it would be well were the specific gravity of the rock given by every engineer who publishes data on any particular kind of rock excavation or mining. Then, too, it often happens that broken rock is purchased by the ton even for civil engineering work, or by the cord of loosely piled rubble for architectural work, thus emphasizing the importance of stating not only the specific gravity but the percentage of voids.

The specific gravity of any material is the quotient found by dividing its weight by the weight of an equal bulk of water. Water, therefore, has a specific gravity of 1; a cubic foot of any substance like granite, having a specific gravity of 2.65, weighs 2.65 times as much as a cubic foot of water. A cubic foot of water weighs 62.355 lbs., or practically 62.4 lbs.; hence a cubic foot of solid granite weighs, $2.65 \times 62.4 = 165.3$ lbs.

When any rock is crushed or broken into fragments of tolerably uniform size it increases in bulk, and is found to have 35 per cent. to 55 per cent. voids or inter-spaces, depending upon the uniformity of the fragments and their angularity. Rounded fragments, like pebbles, pack more closely together than sharp-edged or angular fragments. A tumbler full of bird shot has about 36 per cent. voids, and it is possible to hand-pack marbles of uniform size, so that the voids are only 26 per cent. Obviously, if small fragments of stone are mixed with large fragments the voids are reduced. Pit sand ordinarily has 35 to 40 per cent. voids. Hard broken stone from a rock crusher has about 35 per cent. voids if all sizes are mixed and slightly shaken down in a box; whereas, if it is screened into several sizes, each size has about 45 to 48 per cent. voids.

A soft and friable rock-like shale breaks into fragments having a great range in size, from large chunks down to dust; and, as a consequence, such soft broken rocks have a much lower percentage of voids than the tougher rocks.

The following table shows the swelling of rock upon breaking:

Voids	30%	35%	40%	45%	50%	55%
No. of cu. yds. (loose measure) made by each cu. yd. of solid rock	1.43	1.54	1.67	1.82	2.00	2.22

Hard rock when blasted out in large chunks and thrown into cars or skips may ordinarily be assumed to have from 40 to 45 per cent. voids, hence 1 cu. yd. of hard solid rock ordinarily makes 1.67 to 1.82 cu. yds. of broken, or crushed rock.

Tables of Weights of Rock.—Tables I. and II. will be found useful for computing the weight of solid or broken rock from the specific gravity. Thus, suppose it is desired to ascertain the weight of a solid cubic yard of granite, also the weight of a cubic yard of crushed granite having about 40 per cent. voids. In Table I. it is seen that granite has a specific gravity of 2.55 to 2.86; assuming 2.7 as an average and turning to Table II., we find that a rock having a specific gravity of 2.7 weighs 4,546 lbs. per cu. yd. solid, or 2,727 lbs. per cu. yd. when broken up, so that 40 per cent. of the mass is voids.

TABLE I.

SPECIFIC GRAVITY OF COMMON MINERALS AND ROCKS.

Mineral or Rock	Specific Gravity
Apatite	2.92—3.25
Basalt	3.01
Calcite, $CaCO_3$	2.5 —2.73
Cassiterite, SnO_2	6.4 —7.1
Cerrusite, $PbCO_3$	6.46—6.48
Chalcopyrite, $CuFeS_2$	4.1 —4.3
Coal, anthracite	1.3 —1.84
Coal, bituminous	1.2 —1.5
Diabase	2.6 —3.03
Diorite	2.92
Dolomite, $CaMg(CO_3)_2$	2.8 —2.9
Feldspar	2.44—2.78
Felsite	2.65
Galena, PbS	7.25—7.77
Garnet	3.15—4.31
Gneiss	2.62—2.92
Granite	2.55—2.86
Gypsum	2.3 —3.28
Halite (salt), NaCl	2.1 —2.56
Hematite, Fe_2O_3	4.5 —5.3
Hornblende	3.05—3.47
Limonite, $Fe_3O_4(OH)_6$	3.6 —4.0
Limestone	2.35—2.87
Magnetite, Fe_3O_4	4.9 —5.2
Marble	2.08—2.85
Mica	2.75—3.1
Mica Schist	2.5 —2.9
Olivine	3.33—3.5
Porphyry	2.5 —2.6
Pyrite, FeS_2	4.83—5.2
Quartz, SiO_2	2.5 —2.8
Quartzite	2.6 —2.7
Sandstone	2.0 —2.78
" Medina	2.4
" Ohio	2.2
" Slaty.	1.82
Shale	2.4 —2.8
Slate	2.5 —2.8
Sphalerite, ZnS	3.9 —4.2
Stibnite, Sb_2S_3	4.5 —4.6
Syenite	2.27—2.65
Talc	2.56—2.8
Trap	2.6 —3.0

TABLE II.

Specific Gravity.	Weight in Lbs. per cu. ft.	Weight in Lbs. per cu. yd.	Weight in Lbs. per cu. yd. when Voids are				
			30%	35%	40%	45%	50%
1.0	62.355	1,684	1,178	1,094	1,010	926	842
2.0	124.7	3,367	2,357	2,187	2,020	1,852	1,684
2.1	130.9	3,536	2,475	2,298	2,121	1,945	1,768
2.2	137.2	3,704	2,593	2,408	2,222	2,037	1,852
2.3	143.4	3,872	2,711	2,517	2,323	2,130	1,936
2.4	149.7	4,041	2,828	2,626	2,424	2,222	2,020
2.5	155.9	4,209	2,946	2,736	2,525	2,315	2,105
2.6	162.1	4,377	3,064	2,845	2,626	2,408	2,189
2.7	168.4	4,546	3,182	2,955	2,727	2,500	2,273
2.8	174.6	4,714	3,300	3,064	2,828	2,593	2,357
2.9	180.9	4,882	3,418	3,174	2,929	2,685	2,441
3.0	187.1	5,051	3,536	3,283	3,030	2,778	2,526
3.1	193.3	5,219	3,653	3,392	3,131	2,871	2,609
3.2	199.5	5.388	3,771	3,502	3,232	2,963	2,694
3.3	205.8	5,556	3,889	3,611	3,333	3,056	2,778
3.4	212.0	5,724	4,007	3,721	3,434	3,148	2,862
3.5	218.3	5,893	4,125	3,830	3.535	3,241	2,947

On the other hand, if the weight per cubic yard of the loose broken stone is known (as is often the case), and if the specific gravity has been determined by a test, then Table II. can be used to find the per cent. of voids in the broken stone. Thus, if a given sandstone has been found to have a specific gravity of 2.4, and upon shipping in cars it has been found to weigh 2,200 lbs. per cu. yd., measured loose in the car box, then from Table II., it is seen that about 45 per cent. of the mass of broken stone is voids.

Measurement of Rock.—Sedimentary rock quarried in slabs that are corded up carefully by hand may have 30 per cent., or less, voids, which makes it evident that a contractor in buying rock by the cord should be careful to specify that it be packed closely and not dumped in piles helter skelter before measurement. In buying rock by the "cord" there is another precaution to be taken, and that is to specify how many cubic feet constitute a cord. A cord of wood is $4 \times 4 \times 8 = 128$ cu. ft., but a "cord" of stone is commonly $1 \times 4 \times 8 = 32$ cu. ft. Likewise the word perch, when used, should be clearly defined. A perch of masonry is commonly taken as being 25 cu. ft. (or nearly 1 cu. yd.), but the original perch was a wall 12 ins. high, 18 ins. wide and a rod (16½ ft.) long, making 24¾ cu. ft. In certain localities the "perch" is taken as being only 22 cu. ft. These facts the contractor should know, for he must often deal with quarrymen who will not sell rock by the cubic yard.

Rock is often purchased by the ton of 2,000 lbs.; but to avoid lawsuits it is wise to define the word "ton" in any written or verbal contract, for a ton means 2,240 lbs. in some localities.

If crushed stone for macadam or ballast is purchased by the cubic yard measured loose, the precaution of stating where the measurement is to be made should always be taken. I have made measurements of wagon loads of broken stone after loading from chutes at the bins, and again after traveling for half a mile or more. A surprising shaking down, or settlement, always takes place, ordinarily making a reduction in volume of 10 per cent.

Rock excavation is commonly measured in place before loosening and paid for by the cubic yard of actual excavation; but, in sewer work and in tunnel work, if the contractor excavates beyond certain "neat lines" shown in the blue-prints, no payment is made, unless the specifications explicitly provide for payment for excavation beyond these "neat lines." In trench work, for example, a contractor often has to excavate from 6 to 18 ins. below the grade shown in the blue-print, because it costs less to do so than to work too close to the grade and afterward break off projecting knobs with a bull-point or otherwise. The same is true of shallow excavation, or skimming work, in road construction and the like.

In examining specifications care should also be taken to note whether mention is made of rock slips or falls; for it often happens that after blasting to the neat lines a huge slide of rock occurs, possibly filling the entire excavation. Who is to stand the cost of removing this slide? If it is specified that the contractor shall, then he should study the dip of the rock and its character with this question of sliding in mind.

It is occasionally specified that rock for rip-rap shall be paid for by the cubic yard after deposition; although it is a most unsatisfactory and uncertain way, for much of the rock may be lost in the mud or rolled away by the current. If the rock is delivered in scows it is a simple matter to estimate its weight by water displacement.

CHAPTER II.

METHODS AND COST OF HAND DRILLING.

Kinds of Hand Drills.—Drilling holes in rock by hand may be effected in three ways: (1) By a rotary drill or auger; (2) by a churn-drill; (3) by a hammer-drill, or "jumper" drill, struck with a hammer. A rock auger operated by hand is used only in very soft rock or coal.

A churn-drill, as its name implies, is raised and allowed to drop, or is hurled against the rock. For shallow holes of small diameter it is necessary to give a churn-drill additional weight, which is done by welding a ball of wrought iron to the center of the drill shank, making a ball-drill. A ball-drill is usually provided with a cutting bit at each end and is operated by one man. For deep drilling, that is, for holes more than about 2½ or 3 ft. deep, an ordinary churn-drill is used, operated by one man for shallow work, two men for deeper work and three or even four men for very deep holes where the weight of metal becomes considerable.

The Theory of Drilling.—A hammer-drill, often called a "jumper," although it is not jumped, may be driven by one man, who holds the drill with one hand and strikes it with a hammer in the other hand. In this way holes up to 3 ft. in depth can ordinarily be drilled cheaper than when one man holds the drill while two men strike. But in discussing the relative economy of one-hand drilling as compared with two-hand drilling, authorities appear to have ignored the factor of depth of hole. As the hole grows deeper the advantage of a heavy hammer becomes more and more apparent; but a heavy hammer requires a man's two arms to swing it. A large percentage of the energy of the blow is always consumed in compressing the head of the drill, the shank of the drill and the hammer itself, leaving at best only a small percentage to overcome the cohesion of the rock with the drill bit. In pile driving in soft mud about 65 per cent. of the energy of the hammer is lost in heating the pile-head, etc., leaving only 35 per cent. to overcome the friction of the mud on the pile; and in an analogous way much of the energy of a hammer in the hands of a driller is lost. The longer the pile and the lighter the pile hammer, the greater the loss of energy and the less effective the blow; so in drilling there comes with increased length of drill a decreased percentage of energy that reaches the bit. In fact, if the drill be made long enough, the blow of a one-hand hammer has absolutely no effect in cutting the rock, although a two-hand hammer still has effect.

The churn-drill in the hands of a skilled driller is the most effective type of hand drill for vertical holes; and a little theory is not without its practical value in seeking the reason for the effectiveness of the churn-drill. As before stated, much of the energy of the blow of a hammer is lost in the form of heat at the head of the drill. This loss does not occur with the churn-drill. Moreover, the element of time involved in wedging off a chip of rock also appears to be an important factor. When the cutting edge, or wedge, of a churn-drill first reaches the bottom of the hole, after its descent, the work of wedging off a rock chip begins. A wave of compression travels up several feet of steel before the last ounce of steel in the drill bar has done its work. Obviously it must take about 12 times longer for a churn-drill 6 ft. long to do its wedging work than for a hammer 6 ins. long to do its work. This comparatively gradual and steadily increasing wedging action of the churn-drill theoretically should be more effective than the more sudden action of a hammer-drill; and, as a matter of fact, it is.

It takes some skill to start a hole with a ball-drill and to keep it plumb; but the time spent in acquiring this skill is repaid many times over if quarry operations with hand-drills are to be moderately extensive.

The effect of the size of the hole upon the speed of drilling appears never to have been carefully determined. One authority says that to double the diameter of the hole decreases the speed of drilling by one-half. Another authority thinks that doubling the diameter divides the speed by four. According to the first authority, if a man could drill 12 ft. of 1 in. hole in a shift, he could drill only 6 ft. of 2-in. hole in a shift. According to the second authority, only 3 ft. of 2-in. hole could be drilled per shift. As bearing upon this point the reader is referred to some experiments with different sizes of bits used in machine-drilling tests, an abstract of which appears on page 54.

In drilling by hand it is evident that the blow of a hammer is most effective when directed vertically downward, less effective when directed horizontally and least effective when directed upward. The only careful experiments to determine the relative speeds of drilling at different angles appear to be those of Prof. Hofer. The time required to drill 1 inch of hole in graywacke (a slate, or grit, or conglomerate?) with a hammer-drill, with holes at different angles, was as follows:

85° down (nearly vertical)	152 secs.
60° "	188 "
52° "	241 "
27° "	282 "
2° "	257 "
0° (horizontal)	323 "
24° up	345 "

From which it appears that, at these rates, 16 ft. of vertical hole or 7½ ft. of horizontal hole would be drilled in 8 hrs. This shows one of the several reasons why rock excavation in a tunnel by hand is more expensive than open cut excavation; and it indicates the importance of stating the angle of the drill hole when giving data on hand drilling. Some data given by Jarolimek on drilling dolomitic limestone roughly confirm the above:

60° down	193 secs.
10° up	287 "
45° up	345 "

As bearing upon this point of the angle at which holes are drilled some old data may be quoted from Schoen's "Der Tunnelblau" (1874). Schoen gives the comparative number of cubic yards of material excaved in an open cut and in a railway tunnel, by hand work, 8-hr. shift, and using black powder:

	Cu. yds per man, 8 hrs.	
	Tunnel.	Open cut.
Soft ground	8.5	16.2
Ground loosened with pick	5.3	7.2
" " " gad	3.2	6.5
Quarried rock	1.1	2.6
Quarried and blasted rock	0.64	2.0
Blasted rock	0.34	1.3

In considering the effectiveness of vertical hole drilling it should be remembered that a hole which can be kept partly full of water can ordinarily be drilled faster than a dry hole, and holes that "look up" are necessarily dry, unless, as is rarely done, a jet of water is kept playing into the hole. The water in a down hole takes up and holds in suspension the fine particles of rock, leaving a clean face of rock at the bottom of the hole for the bit to work upon. One authority (Aitken) says that in quarrying trap rock the use of water in drilling reduces the time of drilling a hole by 30 per cent. In certain soft shales, however, up holes are drilled faster than down holes, for the dry powdered shale runs out of the up hole; but in a down hole the shale powder makes a stiff mud with the water and cushions the blow of the bit.

Hammer Drilling.—The common weight of hammer for one-hand drilling is 4½ lbs.; for two-hand or three-hand drilling, 10 lbs. The striking face must be flat or slightly rounding, and smaller than the stock of the hammer. The hole is started on a solid and squared surface, with a short drill, for the longer the drill the less effective the blow. Light blows are struck at first. The bit is turned one-eighth of a revolution after each blow to insure keeping the hole truly circular. But in spite of this precaution most hand-drilled holes are three-cornered, or "rifled." This rifling is not very objectionable in ordinary excavation work, but in quarrying square blocks for masonry it is decidedly objectionable because the rock tends to split in the directions of the three angles of the drill hole upon blasting. How to prevent this rifling will be shown in a subsequent paragraph.

A leather or rubber washer is slipped over the drill and kept close to the hole to prevent splashing of the sludge into the eyes of the drillers. Water is poured into the hole, an operation which is called "tending chuck." The water holds the powdered rock in suspension, forming a sludge which must be removed from time to time. For cleaning out this sludge a scraper or "spoon" is used in shallow holes. A spoon is a ¼ to ½-in. rod provided with a disc at each end, the discs being of different diameters to correspond with the size of the hole at different depths. A spiral hook, or drag-twist, is also used for wiping the hole with hay before charging with black powder. A wooden rod, split or broomed at the end is often used for cleaning out a hole. The broomed end of the rod is dipped into the sludge, and when removed from the hole is struck upon the rock to remove the sludge adhering to the broomed end.

For two-hand or three-hand hammer drilling a bit of 1¼ to 1½ ins. is commonly used for the starter, and the extreme depth of hole is ordinarily not over 6 or 8 ft.; for, as previously shown, the effectiveness of the hammer blow falls off rapidly with increased depth. One man holding the drill and two men striking (three-hand drilling) form the most effective gang for railway tunnel work, also for quarry work where men used to churn drilling are not available.

One-Hand vs. Two-Hand Drilling.—In the metal mines of the West skilled miners ordinarily prefer one-hand drilling when working by contract. The miner holds the drill with one hand and wields the hammer with the other. In very hard rock two-hand drilling is said by Drinker to be slightly (15 per cent.) cheaper than one-hand drilling; but in soft rock he says one-hand drilling is 20 to 30 per cent. cheaper than two-hand drilling. Where one man strikes, while the other man holds the drill, the heavier hammer used makes it possible to drill a larger and deeper hole with economy. Where there is room for one man to hold while two men strike, the economy is greater still. In American railway tunnel work two or three-hand drilling is much more commonly seen than one-hand drilling. Reliable data showing the relative economy of one, two and three-hand drilling, using different diameters and lengths of bit, are not to be found in print, although much to be desired.

Churn Drilling.—For drilling vertical holes, churn drilling is cheaper than hammer drilling. The only exception to this statement is the drilling of plug and feather holes only a few inches deep; but even then a skilful quarryman using a ball-drill will churn down a small hole with greater rapidity than a one-hand hammer driller. Almost any tyro, however, can drill plug holes with a hammer drill; but it takes skill to start a hole with a ball-drill.

For deep holes in soft rocks, like shale, a churn-drill, in the hands of two or more men, is the best hand tool in use. By building a light staging over the hole, as many as six men may operate one drill; three men on the ground and three on the staging; but this is seldom done, except where deep soundings are being made to determine the nature of strata. For breaking up shale for steam shovel work holes 20 ft.

deep are commonly drilled. A pit about 18 ins. deep and 3 ft. in diameter is often dug in the earth overlying the rock, and in the center of this pit the drill hole is started. When the hole is well started the men sit on the sides of the pit with their feet in the bottom. I have never been able to see the philosophy of this pit digging, for a circular wooden stool serves as well as the ground to sit on; and, unlike the pit, it can be used over and over again. Moreover, it is an open question whether churn drillers should be permitted to sit down, for in that position their arms and shoulders do the entire work, whereas when standing their back and thigh muscles aid in lifting the drill. When more than two men are operating a drill it is practically impossible to use the back and thigh muscles; hence the efficiency of each man is greatly reduced when a third man takes hold of the drill, unless the third man stands on a scaffolding above the heads of the two men on the ground. A round drill rod ¾ in. in diameter weighs 1.5 lbs. per ft. of length; a 1 in. rod, 2.65 lbs.; a 1¼-in. rod, 4.1 lbs. per ft., and a 1½-in. rod 5.95 lbs. per ft. It is well to bear these weights in mind when considering the work of churn drilling, for it seems such a little thing to add only ¼ in. to the diameter of drill stock, while in fact the addition adds greatly to the work of lifting the drill. Up to a certain limit, weight is desirable in drilling with a churn-drill; for the drill is not hurled, but allowed to drop freely (unless it is a ball-drill), and does its work by virtue of its weight and velocity. But for every class of rock there is a limit to the weight of a churn-drill beyond which there is an actual loss of efficiency, due to the greater number of drillers required to lift the drill.

Cost of Hammer Drilling.—We have seen that the diameter of the hole, the angle at which the hole is driven and the presence or absence of water in the hole all affect the cost of drilling by hand. We have also seen that the method of drilling with hammer-drills or with churn-drills is an important factor in the cost. Obviously the character of the rock is the most important factor; but unfortunately very few reliable records of cost of drilling in different kinds of rock are to be found. From some observations on hammer drilling, with a 1½ in. starting bit I have found that where one man is holding the drill vertically and two men are striking, the rate of drilling a 6-ft. hole is as follows:

	Ft. in 10 hrs.	Cost per ft., cts.
Granite	7	75
Trap (basalt)	11	48
Limestone	16	33

The cost is based upon a wage rate of $1.75 per 9-hr. day per man; and does not include the cost of sharpening drills, which may be taken at 5 to 8 cts. per ft. more.

I have found that a man drilling plug and feather holes in granite, each hole being ⅝ in. diam. by 2½ in. deep, will average one hole in 5 mins., including the time of cleaning out holes, and the driller strikes about 200 blows in drilling the hole. No water is used in drilling these shallow holes, for the dust is readily and quickly cleaned out with a little wooden spoon. In 8 hrs. of steady work about 100 holes can be drilled, which is about 21 ft. of ⅝ in. hole. But in plug and feather work part of the time is spent in selecting rock, driving the plugs, etc., so that 50 or 60 holes drilled and plugged and feathered is generally counted a fair day's work.

I am indebted to Mr. John B. Hobson for the following data of hammer drilling in a British Columbia mine: Rock was augite diorite and firm red porphyry; starting bit, 1¾ ins.; finishing bit, 1¼ ins.; ⅞ in. steel; holes, 6 ft. deep; 8-lb. hammer. Two miners (one holding drill and one striking) averaged 14.8 ft. per 10-hr. shift. With wages at $2 a day the cost was nearly 28 cts. per ft. of hole.

Mr. Frank Nicholson states that in mining chalcopyrite in magnesian limestone at St. Genevieve, Mo., a day's work for a striker and a holder was 12 ft. of hole drilled. The drills had 1¼-in. starting bits, ⅞ in. octagon steel being used. Wages in 1882 were only $1.35 for miners doing this work, the shift being 10 hrs. long. It cost $6.50 per lin. ft. to drive a 4 × 6 ft. drift, using 40 per cent. dynamite.

In driving the Hoosac Railway Tunnel in 1865 (see Drinker) through gneiss and tough mica schist, 2-ft. holes were drilled in the headings, using 1⅜-in. starting bits, and each driller averaged only 3½ ft. of hole per 10-hr. shift. From 7 to 13 bits were dulled in drilling each hole. Drillers were paid $2.25 a day. Black powder was used in blasting, which accounts for the fact that the holes were large in diameter and shallow.

In driving the Glasgow water works tunnels in 1856-1857 (see Simms), through very hard mica schist, drill holes 20 ins. deep and 1¼ ins. diam. were drilled, a new bit being required for each inch of hole. The time required to drill each hole is stated to have been 1¾. hrs., but no statement is made as to the number of men engaged—presumably two or three on each hole.

Ihlseng states that Swedish iron miners each average 5 ft. of hole a day in medium rock; single-handed drilling; holes 2½ ft. deep; but he does not give the size of bits—a serious omission.

On the Nesquehoning Railway Tunnel (Drinker) driven in 1870 through the conglomerates and shales of Carbon county, Pa., drillers received $2.25 and laborers $2 per 8-hr. shift. The cost of hand drilling was 56 cts. per ft. of hole, of which 6 cts. was for steel and sharpening. Machine drilling in this same material cost 14 cts. per ft. of hole, including repairs to drills, etc., but not including interest and depreciation.

In excavating hard porphyry for the rock-fill dam at Otay, Cal., Mr. W. S. Russell states that a good day's work for three men drilling (one holding and two striking) was 6 to 8 ft. of hole, costing about 80 cts. per ft. of hole drilled. The holes were drilled 20 ft. deep vertically and sprung. This was an unusual depth of hole for hammer drilling, and accounts for the high cost per foot. It shows also how uneconomic is hammer drilling in deep vertical holes compared with churn drilling.

In driving a small (3 × 4½-ft.) tunnel through tough sandstone one driller averaged 4 to 5 holes, each 1½ ft. deep, per 8-hr. shift, using ⅞ in. bit for the starter; and, upon cleaning up, the advance was 1 ft. per shift for one man. Each hole was charged with half a stick of 75 per cent. dynamite.

Cost of Churn Drilling.—I am indebted to Mr. W. M. Douglass, of the firm of Douglass Bros., contractors, for the following data on drilling with churn-drills, for railroad work in western Ohio. Three drillers were used for putting down the first 18 ft. of hole in blue sandstone the first day (10 hrs.), and four men were used for putting down the last 12 ft. of hole, so that it required 70 hrs. of labor at 15 cts. per hr., or $10.50, for a 30-ft. hole, making the cost 35 cts. per ft. In brown sandstone it required 70 to 80 hrs. labor to put down 30 ft. The drill holes were 2¾ ins. at top and 1½ ins. at bottom. Drilling with steam drills in this same stone, holes 20 ft. deep, cost 12 cts. per ft., including everything except interest, depreciation and drill sharpening. The cost of hand drilling agrees very closely with my own records of similar work in Pennsylvania.

Trautwine gives the following rates of drilling 3-ft. vertical holes, starting with a 1¾-in. bit, one man drilling with a churn-drill, shift 10 hrs. long:

Solid quartz	4 ft. in 10 hrs.
Tough hornblend	6 " " " "
Granite or gneiss	7.5 " " " "
Limestone	8.5 " " " "
Sandstone	9.5 " " " "

It should be observed that the holes in this case are shallow (3 ft.), and the diameter (1¾ ins.) is large for such shallow holes, indicating that Trautwine's data applied to rock excavation where black powder was used.

Hand-Drill Bits.—A bit has least work to perform at its center (contrary to Drinker's statements), and this is well shown by the fact that a bit wears most rapidly on its projecting ears. The rock near the center of the hole is struck oftener than anywhere else; hence it is quickly cut away; whereas the rock near the circumference of the hole is not struck so often and is more slowly cut or crushed to pieces. The ears of the bit may project considerably beyond the stock of the drill rod when the rock is soft, because they wear less rapidly and resist breaking off better than in tough, hard rock. The width of the bit may be 30 per cent. to 100 per cent. greater than the diameter of the drill rod, depending upon the hardness of the rock.

For one-hand hammer drilling an octagon steel rod ¾ to ⅞ in. in diameter is commonly used; but ⅝ in. to 1 in. steel may be said to be the limits of size used for one-hand drilling. In comparatively soft rock a ⅝ in. octagon bar may have a 1 in. bit, a ⅞ in. bar, a 1¼-in. bit; and a 1⅝-in. bar, a 3-in. bit, the larger sizes of bits being used only in two-hand hammer drilling or churn drilling. In two-hand hammer drilling a 1¼ to 1½-in. hole for a starter in medium hard rock that is to be drilled to depths of 6 or 8 ft. is common; for it must be remembered that as the hole grows deeper it grows smaller in diameter, due to the continuous wear on the ears of the bits, so that unless a reamer were used it would be impossible to have a uniform diameter of hole except in very soft rock.

The least admissible diameter of hole at the bottom is about ¾ in., where dynamite is used. In the days of black powder a much larger hole was necessary in order to hold enough explosive, although it cost more money to drill the hole; thus in the Hoosac Tunnel the holes were only 2 ft. deep and 1⅜ ins. in diameter at the mouth of the hole. It is a hard rock that will wear the ears of the bit so fast as to reduce the diameter of the hole ⅛ in. in drilling 2 ft., so that a hole 6 ft. deep need not have a diameter at the mouth more than ¾ in. greater than at the bottom even in very hard rock.

The chisel edge of a bit is ordinarily made not straight across, but slightly curved. Different authorities have assigned different reasons for giving this curvature to a bit. Drinker says that in hard rock the curve must be quite flat, but in soft rock it must be very rounding; and his reason is that the wear of the bit being greatest at the center permits of a more rounding form for soft rock. As a matter of fact the wear is never greatest at the center of a bit, and in any case his inference is illogical. Aitken says that the bit is made rounding so as to give greater strength to the ears, and following this reason to its logical conclusion would give us a very rounding bit for very hard rock—precisely the opposite of the form recommended by Drinker. To me it seems apparent that a rounded cutting edge is especially desirable in starting a hand-drilled hole, for it insures effectiveness of the first few blows by concentrating the work upon a small area. Moreover, for the first few inches of the hole, a rounded cutting edge is desirable because any slight tilting of the drill will not mean the concentration of the energy of the blow upon one of the ears, which is the weakest part of the drill and most easily broken. In machine drilling this is not important, but in hand drilling it certainly cannot be overlooked. As corroborating this theory it should be noted that the short drills used for drilling plug and feather holes in granite are invariably made very rounding or convex; whereas in drilling deep holes in sandstone consisting of coarse grains poorly cemented together, a perfectly straight-edged drill is common. A rounded or convex bit cuts a cup-like bottom in the drill hole, which aids in keeping the drill centered; and it is not improbable that the cup acts like a mortar in which the chips from the edges of the hole collect at the center and are more quickly pulverized. The collecting of chips at the center of this stone mortar gives the bit more work to do at its center, which is precisely what should be done in view of the fact that the outer edges would otherwise have most of the work to do.

The wedge-shaped edge of a bit is made as sharp as will hold up without rapid dulling or chipping. In hard rock the bit is made thick near the edge, and with angle of nearly 90°. In soft rock the bit is thin and sharp, with an angle of about 45° between the faces. To this rule there is an exception, for where the rock is a sandstone of rather coarse grains poorly cemented together, a bit that has a blunt edge works fastest. This is due to the fact that the grains of poorly cemented stone are easily broken apart and then the blunt edge crushes them quickly like a pestle in a mortar. A blunt bit is used in drilling small holes in earth, but for a different reason; the blunt bit in earth compacts the earth and crowds it aside, necessitating less cleaning out of the hole.

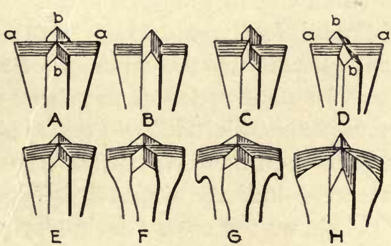


Fig. 1.

Machine Drill Bits.—We have thus far considered only the plain chisel bit, which is the type commonly used in hand drilling. In machine drilling a cross (or square) bit (+), or an ex (×) bit, is commonly used. Occasionally a Z bit is seen. These bits possess the advantage of drilling a more truly circular hole than a chisel bit ever does.

Fig. 1, from an article by T. H. Proske, in the *Engineering and Mining Journal* May 5, 1904, shows shapes of machine drill bits used in the United States. Bit "A" is the common cross (+) bit, and bit "D" is the ×-bit. The theoretical advantage of the X-bit lies in the fact that its cutting edges, aa and bb, are not perpendicular to one another, as they are in the +-bit, so that they do not strike twice in the same place in making one complete revolution; for the bit is turned 1/8 revolution during each up stroke. Bits "E" to "H" are hand-sharpened +-bits, which usually have curved cutting edges, instead of the straight edges of machine-sharpened bits. Bit "B" is the Fitch bit, which has proved very efficient in the extremely hard jaspar of the Champion Mine, Michigan; tests with a 2 7/8-in. Rand drill, under 60 lbs. air pressure, having shown an average drilling speed of 0.28 in. per min. for the +-bit, as compared with 0.66 in. per min. with the Fitch bit. Bit "C" is the Brunton bit, invented by D. W. Brunton, used by the Anaconda Copper Co., Butte, Mont., and at a number of other mines in Montana and Idaho. With this bit the drill piston must turn half way around before the cutting edges strike twice in the same place.

A bit should "mud" freely, and to do this the faces forming the chisel edge should meet at an angle of about 80° for soft rock and 90° for hard rock. A flat chisel edge, like bit "G," or a sharp edge, like bit "H," will not mud freely.

Bit "E" is the common hand-sharpened cross-bit; bit "F" is the same after one-half the originally forged cross has worn off; bit "G" shows the cross nearly worn off; bit "H" is the work of a careless blacksmith, the corners being drawn out and the center not set back.

Sharpening Hand Drills.—A good blacksmith is as essential to economic rock excavation as good hand drillers. For this reason every contractor and every mine manager having charge of drilling operations should know at sight a good blacksmith when he sees him do his work. To be able to do this it is not necessary to become a blacksmith, but simply to learn the art of drill sharpening by reading and by watching and by inquiry. One of the best foremen of rock excavation that I know is a cripple who has never done a stroke of drilling or tool sharpening himself; but he knows exactly how it should be done and cannot be imposed upon by a pretender. The educated man is apt to be fearful of showing himself ignorant of practical work by inquiring into the methods of the drill sharpener.

To begin with the blacksmith must have good drill steel (not tool steel) to work with. Drill steel contains 0.8 to 1 per cent. of carbon. If the steel loses any of this carbon by oxidation it becomes softer and dulls quickly. In heating the bit it is therefore essential: (1) That the heating be not too long continued, nor carried above a cherry red; (2) that the air blast be not too strong; (3) that the bit and some of the shank be well bedded in the coal or charcoal and not in a thin bed of hot cinders. If these rules are not carefully followed the steel will be "burned," which means simply that some of the steel will be oxidized and that this oxide will in turn oxidize some of the carbon of the steel. The heating should be uniform, and to secure uniformity the blacksmith turns the drill over in the fire. When the bit has become a dull cherry red it should be removed with as little delay as possible and dressed. If the corners of the bit are badly worn the chisel edge must first be upset (blunted) to give the proper width; then the drill is held on the anvil at a slope of about 1 ft. rise to 2 ft. horizontal, the edge of the bit being even with the edge of the anvil. In this position it is hammered, turning over at intervals, until a new cutting edge is made. A file may be used (while the bit is still hot) for the final shaping. If the drill is simply dull it is not necessary to upset it, but when taken from the fire it is tapped or brushed to remove any cinders, laid on the anvil and struck with light glancing blows until an edge has been formed. The blows should be glancing so as to draw the steel fibres toward the cutting edge, and the lighter the blows that will accomplish this result the tougher the steel becomes. The width of each bit should be carefully gaged, for nothing is more exasperating to the drillers than to have a careless blacksmith send out bits irregular in width from ear to ear. As above stated, each longer set of drills must have a slightly smaller bit, for all drill holes grow narrower as they grow deeper. The exact reduction in size depends upon the hardness of the rock, and is ascertained by experiment. The bit after being shaped must be reheated for tempering. The heating is done in the forge, as before, until the bit is cherry red, when it is immediately plunged into water for a moment to partly cool it, and then rubbed on a stone to remove the scale, so that the play of colors may be readily seen in a dark corner of the shop. The colors indicate approximately the following temperatures:

Very Pale yellow	430° F
Straw	470°
Brown	490°
Purple	530°
Full blue	560°
Dark blue	600°

As the drill cools the colors should advance parallel to the cutting edge if the cooling is uniform; if otherwise, that side of the bit on which the colors are advancing most rapidly should be held in water. This plunging into the water is sometimes repeated several times before the colors move parallel with the edge of the bit. Finally, when the colors move parallel with the edge, watch the edge closely until it is straw color and plunge into water a short distance, waving it back and forth (to insure rapid cooling) until the steam ceases to form; then leave it in the quenching bath. The quenching bath should be a tub large enough to cool the drills without raising the temperature of the water sensibly. Some of the baths commonly used are brine, water, rapeseed oil, tallow and coal tar; the brine cooling the drill fastest and the coal tar slowest.

Sharpening Machine Drills.—The ordinary + or × bit used in machine drills usually receives treatment somewhat different from that just described, partly due to its shape and partly due to the greater mass of metal in the bit. The

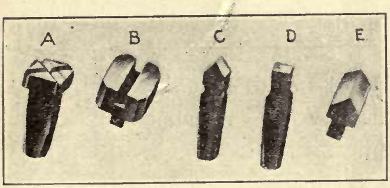


Fig. 2.

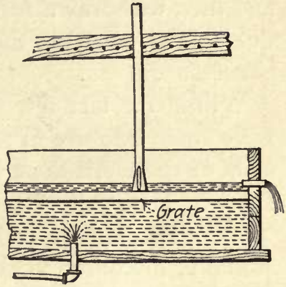


Fig. 3.

bit is first shaped by a special set of blacksmiths' tools, shown in Fig. 2 (from Ingersoll catalogue) consisting of a "dolly," A, "sow," B; "spreader," C; "flatter," D; and "swage," E. The best blacksmith that I have had makes a "dolly" for each size of bit. To do this he heats a block of steel, and drives against it a cold drill bit of the exact shape and gage desired, thus producing what he terms a "female bit," which is afterward tempered hard. The "female bits," or dies, of different sizes are fastened to the anvil so that a hot bit which is to be shaped can be held horizontally and hammered into the die. The result is that all bits are rapidly made true to gage and well shaped. After shaping the bit is reheated for tempering, and at the proper temperature is placed in the cooling bath. Fig. 3 shows a cooling bath (after T. H. Proske), in which a grate or screen is placed ¾ in. below the water surface to support the bit until it is cool. A rack built around the tank, with nails 3 ins. apart, holds the drills upright. The hot steel above the water line prevents the chill from reaching up to the water line, so that only the face of the bit is hardened. The mass of metal in the bit and the fact that at each resharpening the water line is higher up on the drill (due to wear of bit) eliminate danger of cracking at the water line. When this method of cooling is used the edges of the bit should be perfectly straight and not rounding. A bit immersed for a short time, and then withdrawn for annealing, is apt to be soft centered, due to the fact that the center cools more slowly than the corners.

Within the last few years machines for sharpening drills have come into use in some of the large mines. I have records of the work done by two types of machine-drill sharpeners: The "Ajax" and the "Word," which consist essentially of two air-driven hammers, one hammer working horizontally, the other working vertically. Mr. Robert A. Kinzie informs me that the Alaska Treadwell mines use Ajax drill sharpeners, and that one machine sharpens 460 bits per shift. The first Word drill sharpeners were used at the Franklin copper mine, near Houghton, Mich., and at the Black Oak mine, Loulsbyville, Cal. Mr. W. G. Scott, superintendent of the Black Oak mine, is quoted in the *Mining and Scientific Press,* April 11, 1904, as follows:

"The machine ran 183 days with nominal repairs. Average hours run daily, 4; total, 732 hours. One man operated the drill, attended his own forge and made necessary repairs. Any man who can set up and run a machine drill can run the drill sharpener. Approximate number of drills upset and sharpened, 36,000; average, 50 drills per hour. Fuel used is less than one-half that required in hand work. One and one-half minutes are required to form and sharpen a new drill. Over 60 drills have been repointed by this machine in one hour. The life of a bit sharpened by this drill is longer than when done by hand, the bits being better formed and more compact, taking a better and more even temper. The different-sized points are made with uniformity. By a change in the dies the machine will sharpen hand drills. Before we used this machine we employed two drill-sharpening blacksmiths and two helpers to make and sharpen drills. The saving of the machine over hand labor in six months has been $1,738.50; saving on coal (183 days), $183; or a total saving for six months of $1,921.50."

CHAPTER III.

MACHINE DRILLS AND THEIR USE.

Machine Drill Mechanism.—To discuss the mechanical details of rock-drill construction is not within the scope of this chapter, for to do so adequately would require a book in itself. Moreover, a careful study of the line-cuts and descriptive matter of catalogues, supplemented by some machine-shop experience where drills are repaired, is indispensable to any one who wishes thoroughly to acquaint himself with rock-drill mechanism. I shall therefore confine myself to an enumeration of the parts and their functions, referring to Fig. 4, page 33, which is a section of one of the standard makes.

1. The cylinder.
2. Front head of the cylinder and stuffing box through which the piston works.
3. Back head of the cylinder.
4. One of the two side-rods or "through-bolts" that hold the cylinder heads in place.
5. Steam chest.
6. Spool valve.
7. Tappet valve which is oscillated by the shoulders on the piston.
8. The piston. Note the grooves near its ends for the cylinder rings which make an air-tight fit.
9. The chuck at the end of the piston rod for holding the shank of the drill steel.
10. The rifle bar for rotating the piston on each back stroke.
11. Key that aids in holding the shank of the drill steel.
12. The U-shaped chuck bolt with a nut at each end of the U.
13. Two pawls forced by two pawl springs to catch the teeth of the ratchet wheel.
14. The ratchet wheel which prevents the rifle bar (10) turning on the forward stroke of the piston, but allows it to turn on the back stroke.
15. The guide shell on which the cylinder is mounted.
16. The cup by which the guide shell is fastened to the tripod or column arm.

17. Feed nut fastened to the cylinder.
18. Feed screw for moving the cylinder along the guide shell.
19. Crank of feed screw.

Air drills of the rotary type are still in use in Europe; but in America, which is the birthplace of the air drill, no type is in use for rock drilling but the percussive type. The drill is churned back and forth in the hole by compressed air or steam power, and after each stroke it is mechanically turned a fraction of a circle. The drill is fed forward by hand, a crank at the end of a feed-screw being used for this purpose. A longer drill is inserted every 2 ft. in depth of hole, for 2 ft. is the limit of feed of the ordinary feed screw used. Automatic feed devices are not commonly used on drills of ordinary sizes, but only on very large drills for submarine work.

Sizes of Air Drills.—The size of an air drill is denoted by the inner diameter of its air or steam cylinder; thus a 3¼-in. air drill is one having a cylinder 3¼ ins. diam.

The smallest size, 2¼-in. drill, is called a "baby drill," or a one-man drill—the latter name being given to the drill because it can be readily moved about and set up by one man. For narrow work in mines the baby drill is adapted. It is also used largely for drilling plug and feather holes, and might often be used profitably for shallow cuts and trenches. The sizes most commonly used for general contract work, tunneling and mining are the 3⅛-in. and the 3¼-in. drills. A recent report states that in 101 gold mines of the Transvaal, South Africa, 2,355 air drills are in use, and of this number 1,680, or 70 per cent., are 3¼-in. drills. Where the holes are deep and the drilling hard, it is often found that the 3⅝-in. drill is the size to be chosen. Thus, in shaft sinking in syenite at the Treadwell Mine, Alaska, it has been found that the number of feet drilled with the 3⅝-in. drill is fully 30 per cent. greater than with the 3¼-in. drill. As we proceed it will become more and more apparent that the most economic size of drill for any particular

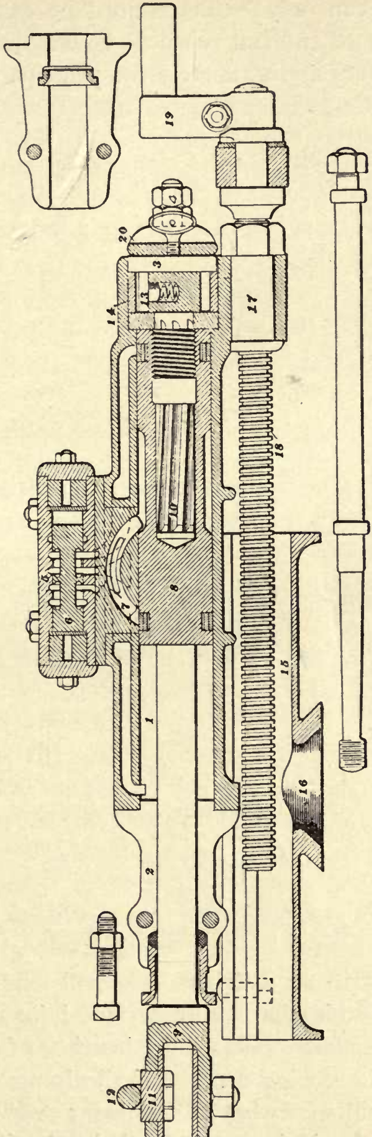


Fig. 4.

class of work can only be determined by experiment, and that as yet no hard and fast rules can be laid down. The following table gives approximately the principal data regarding air drills:

Diameter of cylinder ....ins.	2¼	2½	2¾	3⅛	3¼	3⅝
Length of stroke, ins.	5	6	6½	6⅝	6⅝	7¼
Length of drill from end of crank to end of piston, ins.	36	43	50	50	50	52
Depth of hole drilled without change of bit ....ins.	15	20	24	24	24	24
Diameter of supply inlet (standard pipe) ....ins.	¾	¾	¾	1	1	1¼
Approximate strokes per minute with 60 lbs. pressure at drill	500	450	375	350	325	300
Depth of vertical hole each machine will drill easily, ft.	6	8	10	14	16	20
Diameter of holes drilled as desired from ....ins.	¾ to 1½	1 to 1½	1½ to 2¼	1½ to 2¼	1¾ to 2¾	1¾ to 3
Diameter of octagon steel used ..ins.	¾ to ⅞	⅞ to 1	1 to 1⅛	1⅛ to 1¼	1⅛ to 1¼	1¼-1⅜
Best size of boiler to give plenty of steam at high pressure	6 H. P.	8 H. P.	8 H. P.	9 H. P.	10 H. P.	12 H. P.
Best size of supply pipe to carry steam 100 to 200 feet...........ins.	¾	¾	¾	1	1	1¼
Drill unmounted with wrenches and fittings, not boxed ....lbs.	128	190	265	315	285	390
Tripod without weights, not boxed ....lbs.	80	160	160	160	210	275
Holding down weights, not boxed ....lbs.	120	270	270	285	330	375
Drill, tripod, weights and wrenches, boxed ....lbs.	415	660	820	885	952	1230
Drill unmounted with wrenches and fittings without tripod or column...	$170.00	$200.00	$225.00	$250.00	$275.00	$295.00
Tripod and weights	$30.00	$50.00	$50.00	$50.00	$50.00	$55.00

Handling a Drill on a Tripod.—A drill mounted upon a tripod is the combination commonly used for surface drilling, and even underground on the benches of railway tunnels and often in stoping ores. I shall discuss the handling of the tripod drill somewhat in detail, for every manager of drilling forces should know such of the details as will be set forth, beside many others which the recital of these details may stimulate him to learn for himself by observation.

The order of tripod drilling operations is as follows:

1. Have laborers clean away all earth and loose rock over the sites of proposed drill holes; for earth would clog the drill and would not give a stable support for the tripod. If the surface of the rock to be drilled is loose and shelly, have the laborers clean it away down to solid rock, for a hole cannot be started on loose, shelly rock. I have often seen the expensive drill crew delayed 15 or 20 minutes while one of the crew was occupied cleaning away shelly rock. A laborer at 15 cts. an hour will do this work as well as a drill crew at 50 cts. an hour.

2. Set the tripod over the site of the proposed hole, giving the legs a good spread to secure stability. At best there is considerable vibration when the drill is at work. A small "cat hole" is dug in the rock with a pick or a hand drill for the point of each tripod leg to set in.

3. Having "spotted" the leg points, the legs are adjusted until the saddle is about horizontal. The set screws are then tightened and the tripod leg weights put on.

4. If the machine is not already in its saddle, place it there and fasten the saddle to the cup.

5. Unloosen the nut that clamps the tripod saddle and point the drill in the line of the proposed hole.

6. The "starter," or first drill, is inserted in the chuck after wiping the shank of the drill clean; and the nuts of the chuck bolts are set by first screwing one and then the other until they are perfectly tight. Be sure that the shank of the drill is in as far as it will go before tightening the U-bolt. The starter should have a sharp bit welded to a full sized and perfectly straight drill rod. Beware of a slender drill rod with a nub bit wherever difficulty is expected in starting the hole.

7. The piston is drawn back until it strikes the cylinder head.

The bit is fed forward until the bit strikes the rock, and the point where it strikes is spotted.

9. The piston is shoved into the cylinder, and the bit is raised by the feed-screw.

10. The rock where the bit will strike is faced square for an area of ½ in. or more larger than the bit; for if the drill strikes a glancing blow it may bend the shank, and due to the vibration of the machine, it will vary ½ in. in its alignment.

11. The piston is moved in until it is about the center of the cylinder, and a little oil is let into the cylinder if the machine has not been used in some time, and is operated by air.

12. Turn the air or steam through the hose before coupling it to the drill, in order to blow out any dust or chips; turn off the air and wipe the threads of the coupling clean.

13. Turn off the throttle valve of the machine, then couple on the hose.

14. Let the drill runner test all the nuts with his wrench; and in tightening nuts bear down on the wrench instead of pulling up, as pulling up may shift the machine.

15. Run the bit down to within 1 in. of the rock, for in that position the drill automatically gives a short stroke, and a short stroke is always desirable in starting a hole.

16. Open and then close the throttle if steam is the power used, and work the piston back and forth by hand two or three times, so as to heat everything up evenly and prevent breaking of a part. Then tighten up the side rods which have been left loose to avoid breaking them by the heat expansion. Tighten them evenly and no more than is necessary to secure a tight steam joint.

17. Open the throttle valve part way so that the drill will strike a light blow until a depth of hole has been reached that is greater than the full stroke of the drill, that is, until the bit no longer lifts above the surface of the rock. A little slowness in starting is time saved, because the danger of

breaking the drill is avoided and a true round hole is secured.

18. The helper "tends chuck" by pouring in water, using a can filled from a bucket nearby.

19. When the drill has worked up to its full stroke (6 ins.) the feed-crank is turned slowly so as to keep the bit in position to strike a full blow. If the feed is too slow the piston strikes the cylinder head with a metallic sound that is unmistakable; in which case give the feed-crank a few rapid turns to prevent damage. If the feed is too fast the stroke is automatically shortened, and the rate of penetration of the drill is materially decreased.

20. When steam is used instead of air, more or less steam will condense in the feed pipe during the time that a drill is being moved from hole to hole. Therefore do not let oil into the cylinder until the drill has been running some time (long enough for the water to have all been blown out). The piston must be kept perfectly lubricated to avoid rapid wear.

21. When the feed-screw has reached its limit (the feed is 2 ft. in ordinary sizes), the air is turned off; the drill is raised as far as the feed screw will run, and taken out of the chuck. The hole is cleaned with a "gun" or sand pump. If the hole is shallow a stick broomed at the end may be used to remove the sludge at the bottom of the hole.

These twenty-one rules of ordinary procedure may now be supplemented by a few rules for emergencies and the care of drills.

1. The repeated sticking of a bit in a hole is most exasperating to the drill runner, and the usual remedy is to strike the drill shank viciously with a sledge until the bit comes loose. It is needless to add that this remedy often kills the patient, like other heroic treatments. A moderate blow on the drill shank, near the hole, is a reasonable and often successful means of loosening a stuck bit. A blow should never be hard, and never so high up as to strike the chuck, for a bent piston or a broken chuck is likely to result when hard or high blows are struck.

2. When a bit sticks, nine times out of ten the cause is a crooked hole; and the remedy is a movement of the machine bodily to counteract the tendency of the hole to become crooked. If the drill sticks repeatedly, loosen up the clamp that the shell sets in and determine whether the drill is on line with the hole. If it is not, slacken off on one of the tripod legs so as to throw the drill rod against the side of the hole in the direction the hole is crooking. A lazy driller will hammer his drill; a good driller will reline it.

3. If the bit strikes an inclined layer of rock, and particularly if that layer is harder than the rock above, the bit will glance off toward the "down hill" side and probably stick. The best remedy that I have found in this case is to drop a number of fragments of gas pipe, or other chips of iron, into the hole. These fragments of iron are forced into the soft rock on the lower side, and practically produce a level surface for the bit to strike upon. Old ¾ in., or larger, gas pipe should be cut up into bits with a cold-chisel for this purpose. If the inclined layer of hard rock is not very hard, small quartz pebbles, or the like, will serve instead of iron.

4. In any case, when a drill sticks, shorten the stroke of the drill by feeding down the feed screw, so that the air or steam may get between the piston and the front head; and work for a time with short strokes.

5. Often the cause of sticking is in the bit itself, which may have a broken ear, or the drill rod or shank may not be exactly central with the center of the bit, due to poor blacksmith work.

6. If the rock produces a clay sludge that adheres to the bit and causes sticking, a pipe may be put down in the hole after removing the drill, and a steam or air jet blown through it. This will effectually clean out the sludge when other means fail. Ordinarily, however, all that is necessary is to crank the drill back, pour in a cup of water, turn on about a quarter of the head of air and churn the stiff mud as the machine is cranked up.

7. The softer the rock the more rapidly does the sludge accumulate. To remove the sludge as fast as it forms, a jet of water is most effective. A small pipe is kept in the hole alongside the drill rod, and water is continuously forced through the pipe, either by gravity or by steam or air pressure. In some rocks the increased number of feet drilled per day after a water jet is installed is astounding—amounting often to a 40 per cent. increase.

8. When moving the drill from place to place the piston should be kept inside the cylinder, for otherwise it may be bent if the drill is allowed to fall.

9. A good mineral cylinder oil should be used in the air chest, and from there it passes into the cylinder. Feed in a small amount at frequent intervals, and on a new drill use an excess of oil for the first few days, because the moving parts of a new machine fit tight and hold very little oil at one time.

10. In cold weather, when steam is used, the stuffing box should be unscrewed to let the water out of the cylinder by inclining the drill on its side so as to drain the steam chest and back head.

11. When the machine is not in use it is important to keep the valve and piston well oiled, otherwise rust will rapidly eat away the machined surfaces.

12. Keep on hand a supply of U-bolts and nuts, and have the blacksmith learn to make them, as their life is short at best. A bolt that permits a nut to work loose should be discarded at once, for it is the poorest kind of economy to continue using it. A supply of pawls and pawl springs should also be kept in stock, for while a drill will work with one pawl (after removing the broken one), it will not work economically. Much of the poor work done by drills may be attributed to working with one pawl.

13. In hard rock it often happens that a bit dulls so rapidly that its ears wear off more than is usual; in which case the hole becomes smaller than usual, and, as a conse-

quence, the next new bit will stick on the sides of the hole before reaching the bottom. In this case insert the shank into the chuck, crank up close, turn on the air without tightening the nuts of the U-bolt of the chuck. This will drive the bit straight to the bottom of the hole. Pull it up, turn the bit, and in like manner drive the bit down in a new position; repeat this operation once more and the hole will probably be reamed sufficiently to proceed with the regular drilling.

14. When a drill is choked in the hole and cannot be loosened by hammering, it is often possible to loosen it by running with a loose chuck as just described, and turning the drill with the hands during the back stroke of the piston.

Use of the Column or Bar.—In the early days of tunneling machine drills were mounted on cars running on tracks, and this is still the practice in Europe; but in America the drill is usually mounted on column (Fig. 5) or bar made of 3 to 5½-in. pipe provided with one or two screw jacks at one end. In tunneling the column is usually set upright with blocks of wood between its ends and the rock, although in narrow headings it is often found preferable to set the bar horizontally just as is done of necessity in shaft sinking. The machine is mounted on an arm projecting from the column. The advantage of the column method over the car method of mounting drills is that without waiting for the blasted rock to be entirely cleaned away, the drillers can set up and get to work. A column is preferable to a tripod where it can be used, for it gives a firmer support and there is in consequence less liability of the hole running crooked. Moreover in a stope where the men stand on loose rock it is very difficult to get a solid footing for each of the three tripod legs without laying a substantial flooring of some kind to work upon.

In mining work it is advisable to have an assortment of bars; for one driller may require a bar 3⅛ ft. long for a

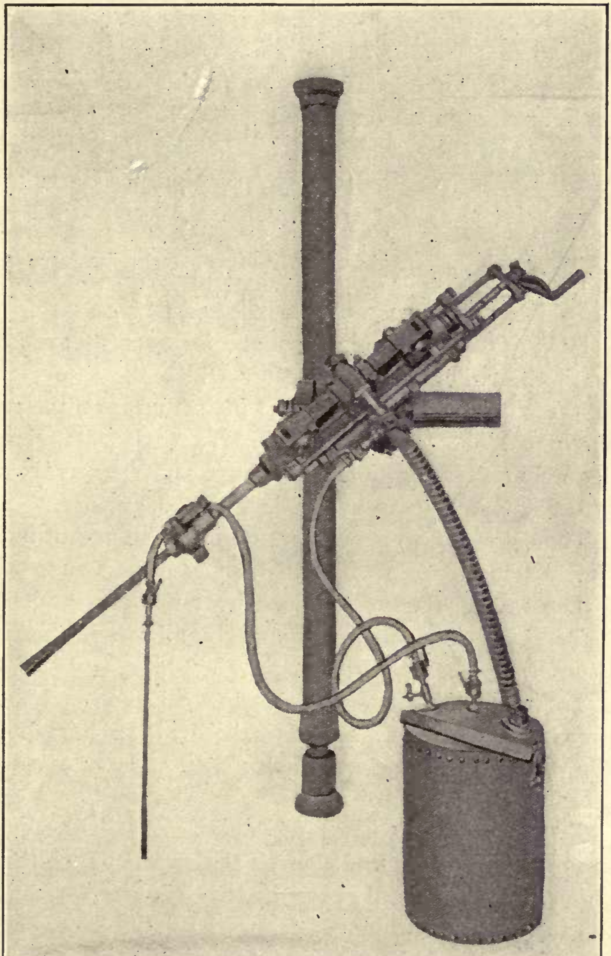


Fig. 5.

horizontal set up, whereas another may find a 9 ft. bar none too long for an upright set up.

1. Blocks of tough wood, reasonably free from knots, are, placed between the ends of the bar and the rock. Sawed wedges about 1 ft. long and of varying thickness at the butt are preferable; but blocks that are flat on the side next to the rock and rounded on the side next to the bar may be used. Most of the blocking should be placed at the jacking end of the bar if possible; a 2-in. piece properly wedged up will serve for the other end, which, in a vertical set up, is the upper end of the bar.

2. The shoe or shoes should fit squarely on the blocking; and to this end the bar may be deflected if necessary.

3. Having placed the blocking, the jack screws are jacked up until the column is solid, after which the safety clamp is put on and its screws set up. If the set up is on the rubble filling of a stope jack up a little at a time, as the rubble settles under the vibration of drilling, and thus avoid splitting the blocking by trying to jack up all at once.

4. The column arm is next put on the column, but its nuts are left a trifle loose, so that the arm may be swung about.

5. The saddle clamp is slipped over the arm and bolted with the clamp side up; and the machine is set in the saddle and swung into line for drilling.

6. To swing the machine so as to drill another hole, the safety clamp is not released until the drill has been pointed in the direction of the new hole; then it is released and clamped in the new position.

7. To dismount the machine, remove the drill steel, release the safety clamp and slacken the arm bolts, so that the machine may be lowered gently by the driller as far as it will go, and the machine removed from its saddle.

8. In starting a hole on a face of hard, slanting rock lower the machine a little on the bar and drill a few inches, then raise the machine and catch the edge of the hole thus started.

Use of Water in Drilling.—A simple and very effective method of increasing the number of feet of hole drilled in soft rock is the use of a water jet to wash the sludge out of the hole as fast as it forms. Fig. 5 shows a drill with a small water pot, pipe and hose equipment for throwing a jet of water into the hole during the drilling. The water pot holds eight gallons and is provided with an air pipe, through which compressed air enters to force the water out through the nozzle. This air pipe is attached to the side of the drill cock. The eight gallons will last an hour or two, depending upon how steadily the machine is running; and a boy can keep a large number of water pots supplied with water. The nozzle may be a piece of ¾-in. gas pipe drawn down to a fine point at the end, and it should be pushed into the hole as the drill advances. In running an adit or drift water may be supplied by gravity from an upper level. $\frac{3''}{4}$!?

Mr. H. P. Stow, of California, is authority for the following data (*Mining and Scientific Press*) showing the effectiveness of a water jet in drilling:

"Three rounds were drilled by the same miner, using a 2¼-inch drill, drilling the same number of hours, size, and as near as possible the holes were of the same kind. Two of the rounds were drilled without taking down the bar, and the third was put in alongside of the other two. He drilled one round without water, one with water, bailing from a bucket, the usual method; and the third with water under pressure in a hose. Without water he drilled 32 ft., using 38 drills; with water by bailing, 41¾ ft., using 33 drills; and with water from the hose, 52 ft., using 37 drills—that is. a gain of 30 per cent. depth of holes, and 50 per cent. gain of feet per drill, with bailing over drilling dry; a gain of 62½ per cent. of depth of holes and 66 2-3 per cent. gain of feet per drill, using the hose over drilling dry; and a gain of 24½ per cent. depth of holes, and 11 per cent. of feet per drill by using hose over bailing water from a bucket. All of which shows that there is not only a gain of ground drilled, but a saving of drill bits used by using water under pressure, instead of bailing it from a bucket or not using it at all. Besides the actual gain in drilling the 'pressure-water' is a saving in getting rid of the gases in the pile of dirt and the dust formed in drilling, resulting materially in the better health of the men, freedom from powder headache and miners' consumption, and increased rapidity of getting into the face to remove the dirt.

"The accompanying illustration, Fig. 5, shows a portable outfit arranged by the Rix Compressed Air & Drill Co., San Francisco, Cal. It consists of a galvanized pot with a bail, holding eight gallons of water, and which, the manufacturer says, has been tested to 150 lbs. per sq. in. pressure. This pot has openings for admitting air pressure on the water, for attaching the water hose, for filling and for releasing the air when filling is required. There are two small pieces of hose, one for admitting the air and one for the squirter. The air hose is attached to the side of the drill cock and lightning couplings are used.

"The superintendent of the North Star mine at Grass Valley, Cal., writes the manufacturers: 'For clear water, we put up small barrels on each level, or in the most convenient place where there is a drip, and run the clear water into this barrel, which is also protected from dust or dirt. A faucet is placed at the bottom. A boy with a heavy galvanized watering pot, filled with a small nozzle, distributes the water to the tanks. A tank of water lasts for some time. It depends on how steadily the machine is running and how careful the man is about shutting off the water when not using it. It will last at least an hour or two anyway, and one boy can keep forty tanks supplied under reasonable conditions of water supply.' "

Concrete is exceedingly troublesome material in which to drill deep holes; but water under pressure has been used very effectively with a wide flare bit which permitted a small copper water pipe to be inserted nearly to the bottom of the hole. The chips and dust were thus carried off by the water before they could wedge the bit, enabling drilling to be done for 25 cts. a ft. at a profit where previously it had been done at a loss.

Other Types of Machine Drills.—The Leyner drill is an American drill especially adapted for use in rock where a water jet enables a bit to cut faster. The Leyner drill has a hollow drill rod through which the compressed air forces the water which escapes through holes near the bit. A small steel water tank having a capacity of 18 gals. is said to hold enough water for a shift's drilling. The compressed air is said to be the principal agent in cleaning the hole, the water laying the dust and assisting in the cleaning. The drill steel is not fastened to the drill piston, and is not churned up and down in the hole, but is struck by the piston, which also rotates the drill bit automatically. This drill, therefore, acts like a hand hammer drill, whereas the common type of drill acts like a hand churn drill.

The Randt rotary drill (see page 333) is a European drill which is driven by hydraulic motors, the water being under great pressure. The bit is held against the rock by hydraulic pressure and is rotated slowly, its cutting teeth chipping off the rock. For tunnel work it has proved effective, but it is not likely to be used for open cutting because of the necessity of forcing the bit against the rock, which would involve loading it with great weights. The pneumatic plug drill (see page 189) is simply a pneumatic hammer used for riveting. It may or may not have a device for automatically rotating the bit. Being a small machine, it is efficient only for drilling shallow holes. I can find no reliable record of its use for holes more than 1 ft. deep. For drilling plug and feather holes, and for block holing boulders and large rocks, this type of drill is destined to have an ever-increasing use. It has already been introduced into several mines and into a large number of dimension stone quarries. Contractors will eventually use it for block holing big rocks in open cut excavation.

The Shaw Pneumatic Tool Co., of Denver, has recently put on the market a pneumatic plug drill with a light column upon which it may be mounted. The drill is held against the rock by air pressure, thus relieving the driller of the strain and jar incident to holding it in his hand. Another feature of the Shaw drill is a small hole through the drill steel, through which the exhaust air passes and forces the dust from the hole, a spray of water being used to lay the dust.

Electric drills (see page 93) have been on the market for a good many years without making much headway in point of numbers in use. It is not improbable, however, that for certain classes of drilling they will eventually find a wide field of usefulness. I reserve for future editions a discussion of their features, trusting that users of electric drills will be kind enough to send me data of actual results under given conditions.

The Automatic Gasoline Rock Drill Co., of San Francisco, has recently introduced a drill operated by gasoline. The explosion of the gasoline in the cylinder drives the piston. Provision is made for taking care of the exhaust and the heat generated by the gasoline explosions. The chief claim made for this drill is that, by its use, steam boilers and compressor plants are dispensed with. This is a type of drill that may prove very economic.

CHAPTER IV.

STEAM AND COMPRESSED AIR PLANTS.

Upon the selection of a power plant for drilling, the profits of rock excavation largely depend. Whether to use compressed air or steam for open-cut excavation is often determined purely by guess work. I have asked several engineers and managers of air compressing plants to explain why it is that compressed air is efficient for drilling in spite of steam engine inefficiency, and invariably the answer has been to the effect that when steam is used direct in the drills there is a great loss of energy in the heat that is constantly radiated along the steam pipe line. One manager said: "It's like trying to heat the wide, wide world with your steam pipe line as the radiator." This sounds plausible, and I doubt not is believed by many to offer a full explanation of the fact that steam operated drills are not economic in the consumption of coal; but that this reason is very far from the truth we shall see presently. Indeed, if the greatest loss of fuel energy came from heat radiated by the steam pipe line, the loss could be practically stopped by the very simple expedient of surrounding the pipes with a lagging of asbestos, hairfelt or the like. The great loss comes from a different source entirely, as will be made clear.

Heat Energy and Horse Power.—The work required to raise 1 lb. to a height of 33,000 ft., or to raise 33,000 lbs. to a height of 1 ft. is 33,000 ft. lbs. (foot-pounds), and if this work is done in 1 minute, it is 1 horse power (1 H. P.) It has been found by experiment that if 778 ft. lbs. of work be expended in churning up 1 lb. of water, the temperature of the water will be raised 1° F. (F. signifies that the common Fahrenheit thermometer is used in measuring the temperature). Hence 1 lb. deg.* = 778 ft. lbs.

Since 1 H. P. = 33,000 ft. lbs., and since 778 ft. lbs = 1 lb. deg., we see that 1 H. P. = 33,000 ft. lbs. ÷ 778 ft. lbs. = 42.42 lb. deg. per min. In a word, if 42.42 lbs. of water are heated 1 degree per minute, the heat energy is exactly equivalent to 1 horse power.

By reference to "steam tables" in any mechanical engineer's hand book it is found that to make 1 lb. of steam at 70 lbs. per sq. in. gage pressure requires 1,146 lb. deg. of heat energy, if the water from which the steam is made is at a temperature of 60° F. to begin with. Now we have seen that 42.42 lb. deg. per min. = 1 H. P.; therefore since there are 60 mins. in an hour 60 × 42.42 = 2,545.2 lb. deg. are equivalent to 1 horse power per hour. Dividing 2,542.2 lb. deg. by 1,146 lb. deg., we have 2.22 lbs. of steam (at 70 lbs. gage pressure from water at 60°) as the equivalent of 1 H. P. That is if one horse power were exerted for an hour in churning up water, the heat developed would be equivalent to making 2.22 lbs. of steam. Reversing the operation, if there were absolutely no losses of any kind, by radiation or otherwise, in the engine, 2.22 lbs. of steam per hour would develop 1 H. P.; but in practice there are many unavoidable losses in the best of steam engines. The exhaust steam itself carries away a tremendous amount of energy that is lost. The following table shows in a striking manner how inefficient the steam engine is at best:

Kind of Engine.	Lbs. of Steam per I. H. P.† per hr.	Steam Efficiency.
Theoretically Perfect (steam at 70 lbs. gage from water at 60°)...	2.22	100%
Compound (good)	15 to 20	14.8% to 11.1%
Single Condensing	23	9.7%
Large Non-condensing (good)....	28	8%
Average Size Condensing.........	30	7.4%
Small Non-condensing	30 to 65	7.4% to 3.4%

* The expression "pound degree" abbreviated to "lb. deg." will be used instead of the common but cumbersome expression, "British thermal unit" (B. T. U.). When 2 lbs. of water are heated so as to raise its temperature 30 degrees, the heat energy imparted to the water is 2 × 30 = 60 lb. degs., which is more in keeping with our system of indicating work in foot-pounds than to say 60 B. T. U.

† I. H. P. is "indicated horse power" as measured with a steam indicator;

When we stop to consider what these figures mean, we wonder how an air compressor run by a steam engine can possibly compete with steam used direct in the drill; for if the heat efficiency of the engine that drives the compressor is only 11 per cent., it means that out of every 100 lbs. of steam only 11 lbs. are utilized to their fullest value, and that 89 lbs. virtually escape into the air without doing any useful work. But even this low efficiency does not mark the end of the losses in an air-compressing plant, for the act of compressing the air raises its temperature, and if the temperature is not kept constant during the process of compression a certain amount of useless work is done by the compressor engine.

The Work of Compression.—If, by means of circulating water, it were possible to prevent the temperature of air from rising during the process of compression, the efficiency of compression would be 100 per cent.; but if the temperature is permitted to rise the air is expanded accordingly and exerts a back pressure upon the engine. Then as the compressed air quickly loses this high temperature in the receiver and in the pipe line, it loses some of its pressure, which represents just so much wasted energy.

It is customary to rate air compressors by the number of cubic feet of "free air" that the compressor will compress to a given gage pressure per minute. By "free air" is meant the ordinary air at sea level and at a temperature of 60° F. The volume of the compressed air as compared with free air is given in the fourth column of Table IV.

The standard practice for large plants now is to compress the air in a low-pressure compressor, pass the air through an "inter-cooler," and then finish the compression to, say, 75 lbs., in a high-pressure compressor. This raises the efficiency of compression to about 84 per cent.

B. H. P. is "brake horse power" as measured with a Prony brake. The B. H. P. of an engine is from 10 to 20 per cent. less than the I. H. P., due to the friction losses in the engine.

Table IV. gives the number of horse power required to compress air under the most favorable and under the worst possible conditions.

TABLE IV.

BRAKE (OR DELIVERED) HORSE-POWER REQUIRED TO COMPRESS ONE CUBIC FOOT OF FREE AIR PER MINUTE TO A GIVEN GAGE PRESSURE. (HASWELL.)

Gage Pressure. Lbs. per sq. in.	B. H. P. Required Under the Worst Possible Condition (Without Cooling).*	B. H. P. Required Under the Best Possible Condition (With Constant Temperature).†	Volume in cu. ft. of Air after Compression at 60° F.
50	.1195	.0951	.2272
55	.1270	.0994	.2109
60	.1342	.1040	.1968
65	.1403	.1081	.1844
70	.1472	.1124	.1735
75	.1537	.1163	.1639
80	.1597	.1193	.1552
85	.1655	.1224	.1474
90	.1710	.1256	.1404
95	.1763	.1289	.1340
100	.1815	.1312	.1281

For the purpose of comparing compressed air with steam Table V. will be found useful:

TABLE V.

STEAM VOLUME AND TEMPERATURE AT GIVEN PRESSURES.

Gage Pressure, Lbs. per sq. in.	Temperature, Fahrenheit.	Lb. Degs. from Water at 32°.	Cu. ft. occupied by 1 lb. of Steam.
50.3	297.8	1,172.8	6.53
55.3	302.7	1,174.3	6.09
60.3	307.4	1,175.7	5.71
65.3	311.8	1,177.0	5.37
70.3	316.0	1,178.3	5.07
75.3	320.0	1,179.6	4.81
80.3	323.9	1,180.7	4.57
85.3	327.6	1,181.8	4.36
90.3	331.1	1,182.9	4.16
95.3	334.5	1,184.0	3.98
100.3	337.8	1,185.0	3.82

Before we discuss the relative efficiency of air and steam it will be best to consider the amount of power required to operate a drill.

---

* Adiabatic.

† Isothermal.

Tests of Air Consumed by Drills.—There are very few reliable tests on record showing the actual air consumption of drills, yet data of some kind must be used by every engineer who has to determine the size of compressor plant needed to operate a given number of drills. The data given in catalogues will not satisfy many engineers. I have, therefore, given considerable space to such records of actual tests as I could find, and to the discussion of the tests compared with catalogue data. By all odds the most satisfactory records of actual air consumption of drills are to be found in a paper by Messrs. J. B. Carper, E. Goffe and W. C. Docharty, recently read before the Mechanical Engineers' Association of the Witwatersrand. Some very interesting data are given of the air consumption of a number of English and American rock drills. The authors state that they endeavored to conduct a fair and unbiased trial of all the rock drills procurable, which represent nearly all of those on the Johannesburg market. The object of the trials was to obtain the quantity of air consumed by the different makes and sizes of drills while doing the same work.

The method of making the tests was as follows: All the holes were drilled in a block of red granite, 2 ft. thick, having the same density and hardness as that of Peterhead granite. The block was bedded in concrete, and the drills were mounted on a quarry bar supported by A frames firmly set in concrete. All holes were drilled vertically. There were two air receivers, 5 × 20 ft. each, having a combined capacity of 757 cu. ft. The air compressor was worked until the gage on the receivers showed a pressure of 80 lbs. per sq. in.; then the stop valve was shut, closing connection with the receiver. Drilling was then begun with the machine to be tested, and continued without intermission until the gage registered 70 lbs. The machine was then stopped, and the depth and diameter of the hole carefully measured. A similar run was made from 70 to 60 lbs., and so on down to 35 lbs. pressure. In working out the results the capacity of the receivers was calculated for every pressure (80 down to 35 lbs.), corrected for temperature, and the air consumption reduced to the equivalent of free air at 70° F. and 24.8 ins. barometer.

The first set of tests was upon 14 drills having 3¼-in. cylinders and using 3-in. bits. Space will not permit reprinting the tabulated results of all these tests but I have calculated the average results of the tests on 13 of the drills, eliminating one drill which behaved in a very erratic manner, and likewise eliminating a few individual runs which showed great departure from other performances of the same make of drill. The following table gives the average results of 13 drills having 3¼-in. cylinders and using 3-in. bits:

TABLE VI.

Air pressure, lbs.	80-70	70-60	60-50	50-40	40-35
Cu. ft. free air per min.	124	117	100	70	60
Cu. ft. free air per lin. in. of hole	95.3	106.4	100.0	116.4	120.0
Cu. ft. free air per cu. in. of hole	13.3	14.8	13.8	15.0	16.6
Lin. ins. drilled per min.	1.3	1.1	1.0	0.6	0.5

In studying this table we see that the air consumption per inch drilled is somewhat erratic, but it was much more erratic in individual tests. The reason for this may be ascribed in part to the lack of uniformity in the granite, in the personal equation of the driller and in the efficiency of the drill itself at different air pressures. Doubtless the tempering and sharpening of the bits had a marked effect also. Each drill, it should be stated, was run by men selected by the agent of the drill, so that the personal equation becomes a factor of importance. However, the personal equation of the driller is by no means of so great importance on short continuous runs like these as upon runs several hours long. In the trials each run was about 6 mins. long; that is, a run of the drill during the drop of 10 lbs. in air pressure consumed on the average about 6 mins.

One thing, however, is strikingly and consistently shown in the above tabulation, namely, the rapid falling off of inches penetrated per minute with every drop in the air pressure. With air pressure averaging 75 lbs. we note that 1.3 ins. per min. were drilled, as against only 0.6 ins. per min. under an average pressure of 45 lbs. The lesson that this teaches is that it does not ordinarily pay to overload an air compressor with more drills than it is designed to carry, and that it never pays to use a main pipe so small as to reduce the air pressure materially.

It is interesting to note that American drills were well in the lead of competitors. The best record of all the 3¼-in. machines was made by a Rand "Slugger." Using a 3-in. bit this drill showed the following results:

TABLE VII.

Air pressure, lbs.	80-70	70-60	60-50	50-40	40-35	80-35
Length of run, mins..	7.166	7,583	8.550	11.666	6.416	41.381
Cu. ft. of free air per min.	91.1	84.9	76.4	52.8	46.1	69.2
Cu. ft. of free air per lin. in. of hole....	61.5	76.9	78.0	96.7	105.3	78.3
Cu. ft. of free air per cu. in. of hole....	8.70	10.80	11.04	13.68	14.90	11.08
Linear inches per min.	1.48	1.10	0.98	0.54	0.44	0.88

The excellence of the performance of this drill is well shown by comparing the results of its test with those above given as the average of 13 drills of the same size. Mr. Docharty attributes the poor showing made by most of the drills to poor valve design resulting in cushioned blows; and he suggests that perhaps a study of these tests will lead some of the manufacturers to improve their valve designs.

There were only four 3-in. machines tested, and, while in some instances the results were erratic, the air consumption and speed of drilling (3-in. bits) correspond very closely with the data above given for the 13 larger drills. It should be remembered, however, that no tests were made in deep holes, for under those conditions the larger machines would doubtless show their superiority. The fact that only shallow vertical holes were drilled should not be lost sight of in considering these tests.

One of the most valuable of the tests made was one with a 2¾-in. Rand "Slugger," using bits of different size. It has been claimed that the depth of hole drilled in a given time should vary inversely as the area of the hole, but there is little in the way of actual tests to substantiate this claim. While too much reliance should not be placed upon one or two tests, the following data tend to prove that in drilling granite the speed of drilling varies inversely as the square of the diameter of the bit. A 2¾-in. "Slugger" with a 6¼-in. stroke gave the following results:

TABLE VIII.

Air pressure lbs	80-71	71-67	60-52	50-44	80-44
Diam. of bit, ins	2 1/16	2	2⅛	2⅛	2 to 2⅛
Length of run, mins	8.08	1.08	9.25	6.83	25.25
Ins. drilled per min	2.19	3.81	1.81	0.95	1.78
Cu. ft. free air per min	72.5	224.2	50.2	47.3	64.0
" " " " " lin. in.	33.0	58.8	27.7	49.7	35.8
" " " " " cu. in.	9.88	18.72	7.83	14.03	10.45

Air pressure, lbs	80-70	70-60	60-50	50-40	40-37	80-37
Diam. of bit, ins	3 1/16	3 1/16	3	3	3	3 to 3 1/16
Length of run, mins	8.75	8.0	10.25	13.67	5.08	45.75
Ins. drilled per min	1.08	0.64	0.69	0.56	0.51	0.70
Cu. ft. free air per min.	69.7	72.0	56.8	43.8	33.2	55.4
Cu. ft. free air per lin. in.	64.2	112.4	81.7	77.3	64.3	78.9
Cu. ft. free air per cu. in.	8.75	15.26	11.56	10.93	9.11	11.09

It will be seen that with a bit of 2 1/16 ins. in diam., the average speed was 1.78 lin. ins. of hole per min.; but with a bit of 3 1/32 ins. in diam., the average speed was only 0.70 lin. ins. per min. A similar series of tests with a Climax machine of the same size gave an average of 1.1 lin. ins. of hole per min. with a 3-in. bit, as against 1.85 lin. ins. with a 2-in. bit. So far as I know these are the only published tests of drilling done under precisely the same conditions with bits of different diameters, and it is much to be desired that further tests be made on different kinds of rock to establish the ratio of speeds using different sized bits in the same drill.

In a rock that makes sludge rapidly as for example shales, slates, and some porphyries, a drill using water under pressure for washing out the sludge will readily excel drills which depend upon the "chuck-tending" of the drill helper. As illustrating this point Table IX shows that the Leyner-Water drill showed up very poorly in comparison with other drills in these tests for air economy; yet this same Leyner-Water drill has made some remarkable records (page 78) in competition with other drills working in softer rocks which make sludge rapidly. Apparently the agents of the drills themselves did not in every case appreciate the difference in results that occur in drilling different kinds of rock, nor do the authors of this valuable paper mention this factor as being one of importance in comparing drill efficiencies.

TABLE IX.

TEST OF LEYNER-WATER DRILL.

(Bore, 3 ins.; stroke, 3 ins.; weight, 156 lbs.)

Air pressure, lbs.	80-70	70-60	80-70	70-60	80-60
Diam. of bit, ins.....	$2^1/_{16}$	$2^1/_{16}$	$2\frac{1}{8}$	$2\frac{1}{8}$	$2^1/_{16}$-$2\frac{1}{8}$
Length of run, mins..	$5^1/_3$	$6^1/_6$	7	$6\frac{1}{2}$	25
Ins. drilled per min...	1.50	1.37	1.21	1.25	1.32
Cu. ft. free air per min.	115.2	100.8	86.9	93.8	98.2
Cu. ft. free air per lin. in.	76.8	73.1	71.6	75.0	74.1
Cu. ft. free air per cu. in.	23.0	21.89	20.2	21.16	21.52

Test of Air Consumption at the Rose Deep Mine.—A 6-hr. run at the Rose Deep Mine, South Africa, showed the following results for 31 drills: The compressed air averaged 70 lbs. per sq. in. and each $3\frac{1}{4}$-in. drill consumed 81 cu. ft. of free air per minute, including all leakage of pipes (there was less leakage than is common in mines). Each drill required 43 lbs. of coal per hour, to supply this compressed air; and each pound of coal developed 3.4 H. P., per hour, by the indicator on the steam engine, evaporating 6.74 lbs. of water from 212° F. The average horse-power of the compressor engine was 12.7 I. H. P. per drill; but all the drillers were trying to make a record and accomplished in 6 hrs. an amount of drilling that ordinarily took 8 hrs. It was an efficient steam power plant, as is seen by the fact that 3.4 H. P. were developed with each pound of coal. The power plant was a vertical King-Reidler Compound Steam Engine and Double Stage Air Compressor with two boilers of the horizontal return tubular type. The engine developed 393 I. H. P. and had a mechanical efficiency of 86 per cent. There were several sizes of machine-drills used, but they were all reduced to the 3¼-in. size as standard by the test of filling the cylinders, ports, etc., with water and ascertaining the volume of water for each drill cylinder. This showed the rating of the drills in air consumption to be as follows:

	Relative Air Consumption
2½-in. drill	0.445
3¼-in.	1.000
3 5-16-in.	1.069
3⅜-in.	1.123

The 31 drills averaged 4.5 ft. of hole drilled per hour for the 6-hr. run; one 3¼-in drill making 52 ft. of hole in 6 hrs., drilling 4 dry holes. Comparing the consumption of 81 cu. ft. of free air per min., at 70 lbs., with the average of 120 cu. ft. (at 70 lbs.) given in Table VI, gives a fair idea of the difference between a long test using a number of drills, and a short test of one drill.

For purpose of comparison I would add that a recent government report states that 2,300 air drills are in use in South Africa, 70 per cent. of which are 3¼-in. drills, and each drill requires 12 I. H. P. of steam engine to run the two-stage air compressors that supply the air.

Tables of Air Consumption in Catalogues.—Table X is given in the catalogue of one of the well-known drill manufacturers, and is said to be based upon actual tests of single drills running continuously without stops for changing bits, etc.

TABLE X.

CUBIC FEET OF FREE AIR PER MINUTE REQUIRED TO RUN A ONE-DRILL PLANT.

Gage Pressure.	Diameter of Drill Cylinder.												
	2"	2¼"	2½"	2¾"	3"	3⅛"	3 3/16"	3¼"	3½"	3⅝"	4¾"	5"	5½"
60	50	60	68	82	90	95	97	100	108	113	130	150	164
70	56	68	77	93	102	108	110	113	124	129	147	170	181
80	63	76	86	104	114	120	123	127	131	143	164	190	207
90	70	84	95	115	126	133	136	141	152	159	182	210	230
100	77	92	104	126	138	146	149	154	166	174	199	240	252

When more than one drill is to be supplied from the same air compressor the manufacturers advise multiplying the quantities given in Table X by the factors given in Table XI.

TABLE XI.

Number of drills....	1	2	5	10	15	20	30	40	70
Multiply value in Table X by......	1	1.8	4.1	7.1	9.5	11.7	15.8	21.4	33.2

Tables similar to these are given by other manufacturers. In answer to letters of inquiry I have been informed that such tables are "based upon experience in a large number of mines." In view of the actual tests above given, tables in catalogues should be used with caution. Certainly the "experience" in all mines is not the same, and, if it were, it would differ widely from "experience" in open cut work where drills usually work more continuously.

In the next chapter it will be shown that the actual drilling time, that is, the time when the drill is actually striking blows, is seldom over 70 per cent. and often not more than 40 per cent. of the length of the shift. Knowing the conditions of work, the reader will be able (with the aid of data given in the next chapter) to predict approximately the per cent. of actual drilling time. Then if there are more than, say, 10 drills, he can multiply the air consumption of one drill (when actually drilling) by the percentage of drilling time in the shift, and the product will be the average air consumption of each drill. If there are less than about 10 drills it will not be safe to figure so closely, because the fewer the drills operated from one compressor, the more likely it is that, all or nearly all, of them will be using air at the same time. The larger the number of drills, on the other hand, the more certain it is that some will be changing bits while others are drilling, and thus draw a steady, average amount of air from the compressor.

Steam Consumption in Terms of Air Consumption.—When steam is piped directly from the boiler into a drill, practically the same number of cubic feet of steam are consumed as of cubic feet of compressed air.* Referring to Table V we find that 1 lb. of steam at 75 lbs. gage pressure occupies 4.8 cu. ft., or 1 cu. ft. steam weighs 0.21 lb. Referring to Table IV. we find that 1 cu. ft. of free air is equivalent to 0.1639 cu. ft. of compressed air at 75 lbs. pressure, or 1 cu. ft. of compressed air (at 75 lbs.) is practically equal to 6 cu. ft. of free air. We may assume that a cubic foot of steam will do practically the same work in a drill as a cubic foot of compressed air at the same pressure, because neither the steam nor the air acts to any great extent expansively in a drill cylinder, due to the late cut-off. This being so, 0.21 lb. of steam is equivalent to 6 cu. ft. of free air, or 1 lb. of steam is equivalent to nearly 30 cu. ft. of free air, or 1 cu. ft. of free air is equivalent to 0.035 lbs. steam—all at the same pressure of 75 lbs. per sq. in. If a drill consumes at the rate of 100 cu. ft. of free air per min., it will consume 6,000 cu. ft. of free air in an hour. If it were using steam in its cylinder instead of air (at 75 lbs. pressure), it would, therefore, consume $6{,}000 \times 0.035 = 210$ lbs. of steam (at 75 lbs. pressure) in an hour. Referring to Table V., we see that 1

* The chief engineer of the Rand Drill Co. informs me that he estimates 10 per cent. less volume of steam than of compressed air due to the fact that steam passes with less velocity through the ports.

lb. of steam (75 lbs. pressure), made from water at 32°, contains 1,179 lb. degs. of heat; but, as the feed water is ordinarily hotter than 32°, we may say that 1 lb. of steam contains 1,150 lb. degs. of heat energy imparted to it by the coal. Therefore, if a steam drill were to consume 210 lbs. of steam in an hour it would use 210 × 1,150 = 241,500 lb. degs. of heat energy.

From Table IV. we find that to compress 1 cu. ft. of free air per min. to 75 lbs. pressure requires 0.1163 H. P.; but we have seen that 1 H. P. = 42.42 lb. degs.; hence 0.1163 × 42.42 = 4.933 lb. degs. per min. are required to compress 1 cu. ft. of free air to 75 lbs. gage pressure. If the air drill consumes 100 cu. ft. of free air per min., we have 100 × 4.933 = 493.3 lb. degs. per min., or 493.3 × 60 = 29,598 lb. degs. per hour. Now comparing these 29,598 lb. degs. in the hour's work of the drill using air, with the 241,500 lb. degs. in the hour's work of the same drill using steam, we see the true reason why compressed air can compete with steam in spite of all the losses of power involved in producing the compressed air. The ratio of 29,598 to 241,500 is practically 1 to 8. In other words, 1 cubic foot of steam, at 75 lbs. gage pressure, contains eight times as much heat energy as one cubic foot of air at the same pressure, yet so far as running the drill is concerned the air is exactly as valuable as the steam. If window weights were made of gold instead of cast iron they would not be one whit more effective in counterbalancing the weight of the window, so steam is not more effective than compressed air when both act directly at pressures that are identical.

Going back again to the steam engine and the compressor it should not be forgotten that they have a combined efficiency of not much more than 10 per cent.; hence although an air drill uses only 29,598 lb. degs. of heat energy per hour, it required 10 × 29,598 = 295,980 lb. degs. per hr. of energy in the form of steam that entered the compressor engines to produce the compressed air supplying the drill. Comparing this 295,980 lb. degs. of energy with the 241,500 lb. degs. consumed by the drill using steam direct we see that, if there is no loss of heat energy by radiation in the steam pipe line, it takes about 25 per cent. more coal to run each drill when compressed air is used than when steam is used in the drill. Ordinarily, however, steam pipes are left bare, and the heat radiated is nearly sufficient to equalize the coal consumption.

The Efficiency of a Steam Pipe Line.—When steam is passing through a wrought-iron pipe there is a constant loss of heat which has been found by experiment to be about 750 lb. degs. per sq. ft. of pipe surface per hour, when the surrounding air is still; and about 30 per cent. more when a wind is blowing. This is upon the assumption that the difference of temperature between the steam and the outside air is 250° F. If the difference in temperature is greater the loss of heat by radiation is proportionately greater. In calculating the number of square feet of pipe surface, bear in mind that the outer surface of the pipe is meant. The results of some experiments given under "Heating" in Encyclopedia Brittanica indicate that cast-iron pipe radiates the heat about 50 per cent. faster than wrought-iron pipe.

TABLE XII.

LOSSES BY FRICTION AND RADIATION IN DELIVERING 1,000 LBS. OF STEAM PER HOUR THROUGH A BARE WROUGHT IRON PIPE 100 FT. LONG, TERMINAL GAGE PRESSURE 75 LBS.

Nominal Inside Diam. of Pipe in Ins.	Lbs. of Steam Lost per Hr. per 100 Lin. Ft. By Friction.	By Radiation.	Total.
1	177.7	22.9	200.6
1¼	58.2	29.0	87.0
1½	23.4	33.2	56.4
2	5.6	41.4	47.0
2½	1.8	50.1	51.9
3	0.7	61.1	61.8
3½	0.3	69.8	70.1
4	0.2	78.5	78.7

NOTE.—The loss due to friction varies as the cube of the number of pounds of steam per hour. Hence divide the delivery steam in lbs. per hr. by 1,000 (1,000 being the basis of the table), cube the quotient and multiply the quantities in column two thereby. Thus if a 3-in. pipe must deliver 2,000 lbs. of steam per hour, we have, 2,000 ÷ 1,000 = 2, and cubing this 2 we have 8, which multiplied by the 0.7 (in column two opposite 3-in.) gives 5.6 lbs. of steam lost per hr. per 100 ft. of 3-in. pipe due to friction. The loss by radiation is the same, regardless of the velocity of the steam in the pipe.

TABLE XIII.

Abstracted from an excellent paper on "Tests of Steam Pipe Coverings," by Geo. H. Barrus, in *Trans. Am. Soc. M. E.*, 1902.

Pipe Covering.	Inner Diameter of pipe, ins. (nominal).	Thickness of pipe covering, ins.	Weight of pipe covering per lin. yd., lbs.	Cost of pipe covering per lin. ft. of pipe, not including labor, cts.	Cost per sq. ft. of pipe surface for covering in place, cts.	Lb. degs. or (B. T. U.) radiated per hr. per sq. ft. of pipe surface: steam at 70 lbs.; outside air at 66°.
N. Y. Air Cell (asbestos)	2	1	3.22	12.32	26	187
Gast's Air Cell (asbestos)	2	15/16	3.77	10.56	23	198
Carey's Moulded	2	1	5.77	8.64	20	192
Asbesto-sponge	2	1	7.25	8.64	..	194
Asbestocel	2	7/8	5.10	9.60	22	182
Magnesia	2	1	3.17	.....	40	143
Asbesto-sponge (48 lams.)	2	1	6.00	19.20	37	143
Asbestos, Navy brand...	2	1 1/8	3.69	18.24	35	151
Asbestos, Navy brand...	10	1 3/8	12.60	66.0	25	97
Magnesia	10	1 3/16	17.88	62.7	24	88
Asbesto-sponge felt	10	1 3/8	28.55	73.3	28	70
Asbesto-sponge felt	10	1 3/16	24.47	54.0	21	75
Bare iron pipe..........	2	....	.....	....	..	750

According to Mr. Barrus, the number of lb. degs. (or B. T. U.) of heat radiated through a pipe covering is inversely proportional to the thickness of the covering raised to the 5/8 power; according to H. G. Stott, the heat radiated is inversely proportional to the square root of the thickness of the covering. The heat losses, given in the last column of Table XIII., are for a difference of temperature of 250° F. between the steam and the outside air; for any other difference in temperature the heat loss is proportional to the ratio of the differences in temperature. The heat losses are given per square foot of the outside surface of the pipe; but to be truly scientific they should be given per square foot of the outside surface of the pipe covering. However, for the tests of pipe covering made on a 2-in. pipe, the results show a slightly higher heat loss than would occur on larger pipes, so that the tabular data are on the side of safety.

Since, according to Table V., 1 lb. of steam of 75 lbs. pressure contains 1,179 lb. degs. of heat energy from water at 32°, or 1,151 lb. degs. from water at 60°, we have simply to divide the loss of heat expressed in lb. degs. by 1,150 to get the number of pounds of steam lost in a pipe line per hour. At 70 lbs. gage pressure the loss is 750 lb. degs. per sq. ft. of bare pipe surface; hence $750 \div 1,150 = 0.66$ lbs. of steam per sq. ft. per hr. Roughly speaking, therefore, an uncovered pipe loses 2-3 lb. of steam per sq. ft. per hr. The following table gives the area in square feet of 100 ft. of pipe:

TABLE XIV.

Nominal inside diam., ins.	Sq. ft. of outside surface of 100 lin. ft. of pipe.	Nominal inside diam., ins.	Sq. ft. of outside surface of 100 lin. ft. of pipe.	Nominal inside diam., ins.	Sq. ft. of outside surface of 100 lin. ft. of pipe.
½	22.2	2½	75.2	6	173.3
¾	27.5	3	91.7	7	198.0
1	34.4	3½	104.7	8	225.2
1¼	43.5	4	117.8	9	250.0
1½	49.8	4½	130.7	10	281.7
2	62.1	5	159.0	..	.....

Since the heat loss in uncovered pipe is about 2-3 lb. of steam per hr. per sq. ft. of pipe surface, we have merely to multiply the number opposite the pipe of given size in Table XIV. by 2-3 to determine the approximate loss of steam. In a ¾-in. pipe the area is 27.5 sq. ft. per 100 lin. ft. of pipe; hence the steam loss is 2-3 $\times$ 27.5 = 18.3 lbs. of steam per hour. If the pipe is 150 ft. long the loss is 27½ lbs. of steam per hour, due to radiation, at least 80 per cent. of which loss can be saved by using a pipe covering little more than an inch thick. Pipe coverings can be bought in short lengths that slip like sleeves over the pipe. For outdoor use the covering should be of some flexible variety, preferably of asbestos fibre (not moulded hard with plaster), wrapped with waterproofed canvas. A fair idea of the prices may be had by inspecting Table XIII.

In a compressed air pipe line there is no loss of energy by heat radiation; hence the larger the pipe the greater its efficiency in conveying the air without reducing its final pressure at the drill. But in a steam pipe line the larger the pipe the greater the loss by heat radiation, whereas the smaller the pipe the greater the loss by friction. Hence for any given quantity of steam to be delivered per hour there is just one size of pipe that will give a minimum loss of energy, which may be calculated from the data given in this chapter.

Flow of Air Through Pipes.—Tables XV. to XVII. are based upon D'Arcy's formula. Table XV. gives the number of cubic feet of free air delivered per minute through a pipe 100 ft. long without any loss of pressure. Table XVI. gives the factors, F., to be used for different lengths of pipe. Table XVII. gives the multipliers, M, to be used to determine the loss of pressure. The following formulas are to be used with these tables:

(1) $Q = L \times F \times M$ (3) $L = \dfrac{Q}{F \times M}$

(2) $M = \dfrac{Q}{L \times F}$ (4) $F = \dfrac{Q}{L \times M}$

Q = cu. ft. of free air discharged per min.

L = factor given in Table XV.

F = factor given in Table XVI.
M = " " " " XVII.

Example I. Given a 4-in. pipe, 600 ft. long, initial air pressure 60 lbs., required to discharge 1,200 cu. ft. of free air per min., what will be the terminal pressure?

By Table XV., under 4-in. pipe and opposite 60 lbs., we find L = 1,535.

By Table XVI., for 600 ft., F = 0.408.

$$\text{Hence, } M = \frac{Q}{L \times F} = \frac{1,200}{1,535 \times 0.408} = 1.9 \text{ lbs.}$$

Now by Table XVII., opposite 60 lbs. pressure and under 4 lbs. reduction, we find M = 1.89, so that the loss of pressure being 4 lbs. we have a terminal pressure of 56 lbs.

Example II. Given a 6-in. pipe, 2,000 ft. long, initial pressure 80 lbs., terminal pressure 70 lbs., what will be the volume discharged?

By Table XV., under 6-in. pipe and opposite 80 lbs., we find L = 4,971.

By Table XVI., for 2,000 ft. length, F = 0.224.

By Table XVII., under 10 lbs. reduction and for 80 lbs. pressure, we find M = 2.82.

$Q = L \times F \times M = 4,971 \times 0.224 \times 2.82 = 3,140$ cu. ft. per min.

TABLE XV.

GIVING FACTOR L.

Gage, Lbs.	Nominal Diameter of Pipes in Inches.											
	1"	1½"	2"	2½"	3"	3½"	4"	5"	6"	7"	8"	10"
50	38.96	122.4	243.4	396.3	701.5	1030	1430	2558	4114	5993	8312	14910
60	41.83	131.4	261.1	425.4	752.9	1105	1535	2747	4416	6433	8920	16000
70	44.53	139.9	278.0	452.9	801.8	1176	1634	2925	4701	6848	9499	17030
80	47.08	147.9	294.0	478.8	847.6	1244	1728	3091	4971	7240	10040	18000
90	49.54	155.6	309.3	503.8	891.8	1307	1817	3253	5230	7619	10560	18940
100	51.88	163.0	324.0	527.5	933.8	1370	1904	3407	5477	7979	11050	19850
110	54.10	169.9	337.8	550.1	973.9	1429	1985	3552	5712	8320	11530	20690
125	57.15	179.5	356.8	581.3	1028	1510	2097	3754	6034	8789	12180	21860
150	62.10	195.1	387.8	631.7	1117	1641	2280	4080	6558	9553	13240	23760

TABLE XVI.

GIVING FACTOR F.

Length, feet.	Multiplier F.	Length, feet.	Multiplier F.
100	1.0	6000	0.129
200	0.707	7000	0.119
300	0.577	8000	0.112
400	0.500	9000	0.105
500	0.447	10000	0.100
600	0.408	12,000	0.0912
750	0.365	15,000	0.0817
1000	0.316	......	......
1250	0.283	......	......
1500	0.258	.....	.....
2000	0.224	......	......
2500	0.200	......	......
3000	0.183	......	......
3500	0.169	......	......
4000	0.158	......	......
5000	0.141	......	......

TABLE XVII.

GIVING FACTOR M.

Initial Gage Pressure. Pounds.	Reduction of the final pressure in pounds per square inch. 1	2	3	4	6	8	10	12	14	16	18	20
50......	0.984	1.37	1.65	1.87	2.22	2.48	2.67	...	...	...	...	...
60......	0.986	1.37	1.66	1.89	2.24	2.52	2.74	...	...	...	...	...
70......	0.988	1.38	1.67	1.90	2.27	2.56	2.79	2.97	3.12	3.24	...	...
80......	0.989	1.38	1.67	1.91	2.29	2.59	2.82	3.02	3.19	3.32	3.43	3.53
90......	0.990	1.38	1.68	1.92	2.31	2.61	2.86	3.06	3.24	3.39	3.51	3.61
100......	0.991	1.39	1.68	1.93	2.32	2.63	2.88	3.10	3.28	3.44	3.57	3.69
110......	0.992	1.39	1.69	1.93	2.33	2.64	2.90	3.13	3.32	3.48	3.63	3.75
125......	0.993	1.39	1.69	1.94	2.34	2.66	2.93	3.16	3.36	3.54	3.69	3.83
150......	0.994	1.39	1.70	1.95	2.36	2.69	2.97	3.21	3.42	3.61	3.77	3.92

Fuel and Steam Boiler Efficiency.—Steam boilers are commonly rated as having so and so many horse-power capacity. This is a very misleading and unsatisfactory way of rating a boiler, for the horse power of work that a boiler can do depends entirely upon the kind of engine to which the steam goes. A boiler that supplies 1,600 lbs. of steam per hour to compound condensing engine using 16 lbs. of steam per H. P., evidently develops 100 H. P.; yet if this very same boiler is made to feed an ordinary single cylinder non-condensing engine using 40 lbs. of steam per hour, it will develop $1600 \div 40 = 40$ H. P. In buying a boiler, therefore, be sure to secure the manufacturer's guarantee, not of its horse power, but of its steam capacity in pounds of steam per hour at a given gage pressure (say, 70 lbs. per sq. in.), using a fuel of a kind stated in the guarantee. The American Society of Mechanical Engineers has recommended that wherever the word horse power is used in reference to boilers, it shall mean 30 lbs. of water evaporated from 100° F. to steam having a gage pressure* of 70 lbs. per sq. in. under average firing without forcing the boiler, and by forcing the same boiler should be capable of evaporating 1-3 more steam per hour than its ordinary rating; that is, a 100 H. P. boiler (30 lbs. steam per hr. per H. P.) should be capable of developing 133 H. P. if forced. Unfortunately manufacturers usually pay no attention to this suggested rating, and perhaps they can hardly be blamed, because if it were always followed we should see at times a 100 H. P. boiler used to run a 200 H. P. compound condensing engine, while at other times the same 100 H. P. boiler would be used to run a 100 H. P. non-condensing single engine.

By careful tests it is easy to ascertain how many lb. degs. can be developed by 1 lb. of any fuel burning under perfect conditions. Thus 1 lb. of perfectly pure carbon will develop 15,000 lb. degs.; that is, it will raise the temperature of 15,000 lbs. of water 1°, or 150 lbs. of water 100°. Coal is never entirely pure carbon, but contains some ash and other materials. All mechanical engineers' handbooks contain tables of the heating value of different kinds of fuel, with which it is well to be familiar whenever there is a choice of fuels.

When coal is burned under a boiler a large percentage of its heat passes up the chimney in the gases and is lost; and in addition to this loss the boiler itself radiates heat constantly. The greater part of the loss occurs in the heat that

* The atmosphere has a pressure of 14.7 lbs. per sq. in., and since a steam gage shows the steam pressure above atmospheric pressure, we must add 14.7 to the gage pressure to get the "absolute pressure."

goes up the chimney. In large, well designed boilers, properly protected by asbestos or similar covering, the coal burned will develop steam to about 85 per cent. of the full heat value of the fuel; the efficiency of the boiler and furnace is then 85 per cent. In locomotive boilers where forced draft is used, firing not of the best and boiler exposed to moving air, the efficiency is often as low as 45 per cent. The efficiency of a good boiler of moderate size (100 H. P.), well housed, is ordinarily about 75 per cent. A small (20 H. P.) boiler exposed to the wind has an efficiency of 55 to 60 per cent. when not forced.

If a small boiler is used to run one drill, the boiler must always have up enough steam to keep the drill running at nearly full capacity; but when the drill is stopped, during the changing of bits, moving, etc., there is a waste of steam, because the period of stoppage is not long enough to permit the fireman to make any material change in the firing and in the draft. Hence one single drill (3¼ in.) must be counted upon as using about 250 lbs. of steam per hour (see page 58). A 1-in. steam pipe 200 ft. long, if not covered, will lose nearly 50 lbs. of steam per hour by condensation (and often as much more by leakage), thus making a total steam consumption of $250 + 50 = 300$ lbs. per hour, due to the drill and the pipe line. If 1 lb. of the coal will develop 14,000 lb. degs., and if the small exposed boiler has an efficiency of 58 per cent., we have $14{,}000 \times 58$ per cent. $= 8{,}120$ lb. degs., which divided by 1,150 (the lb. degs. required to produce 1 lb. of steam at ordinary pressures), gives about 7 lbs. of steam produced by 1 lb. of coal. Therefore, $300 \div 7 = 43$ lbs. of coal required per hour to supply steam for the drill and pipe line. We have still to add the loss of fuel when the fire is drawn at night, as well as the loss of radiated heat during the starting of the fire in the morning, and leakage losses. Not less than 150 lbs. of coal per day are thus consumed in a small boiler, bringing the total coal consumption up to 580 lbs. of coal for running the one drill one 10-hr. shift. Where two drills are run by steam, the consumption of coal per drill is somewhat less. Where a large number of drills are in use, all are seldom running at the same time, but at no time is the boiler entirely without some drills drawing steam from it. Under such conditions, and where the main pipe line is lagged, each drill will require about 250 lbs. of coal per 10-hr. shift under average conditions. Knowing the number and size of drills, the size of steam pipes, the character of lagging around the pipes, the depth of drill holes and kind of rock, it is possible to estimate the coal consumption per drill with considerable accuracy by applying the data given in this chapter and in the next chapter.

Merits of Compressed Air.—A compressed air plant was recently installed at a large stone quarry where drills and channelers had formerly been run by steam direct from a large number of small boilers. When the compressed air plant was installed the coal consumption was reduced from 50 tons per day to 15 tons per day. Due to the higher and more even pressure of the air (as compared with the steam from the small boilers), fewer drills and channelers were needed, because each did more work than before. Moreover, there were no delays in the morning, getting up steam, as is usually the case where a large number of small boilers are operated. This excellent and remarkable result could have been accomplished at less expense by installing a central steam boiler plant and using lagged steam pipes. The steam pipes would have required expansion joints and traps for draining off water of condensation similar to those in the air pipe system. This plant was widely advertised as proving conclusively the advantage of using compressed air instead of steam in drills. What it really proves is that a central power plant is far more economic than a large number of small plants. We have seen that the efficiency of small boilers is often 45 per cent. or lower, as compared with the 85 per cent. efficiency of large boilers. We have also seen that a small boiler supplying one or two drills must always carry enough steam to keep both drills going at their full steam consumption, in spite of the fact that the drills are not working more than 40 to 70 per cent. of the time; whereas with a large central plant the drills "average up," some running while others are not, thus greatly reducing the average daily coal consumption. Compressed air has many advantages over steam for operating drills, but a reduction in the coal bill is not one of them (in spite of the general belief to the contrary) if a fair comparison is made between a central steam plant with covered pipes and a central compressor plant.

Compressed air, however, possesses several advantages distinctly its own, which may be enumerated as follows: (1) It does not rot the hose from the pipe to the drill, and a much cheaper hose may be used and consequently a longer hose than with steam. (2) Less oil is required to keep the drill lubricated. (3) In warm weather the exhaust air makes working around the drill comfortable. (4) A trench or quarry pit is not filled with steam, making it difficult at times to see. (5) There is no danger of injuring the drill itself by sudden expansion due to heat. (6) There are no pipes to thaw out in winter due to condensed water carelessly allowed to collect. (7) Plug drills can be used for block holing, plug and feathering, etc. (8) The air can be used for blowing the sludge and water out of a hole before charging. (9) The air can be used for forcing a jet of water into a hole alongside of the drill-rod. (10) The machine being cool is easily handled. (11) Water power may be used for compressing the air.

Efficiency of the Jerome Reservoir Plant.—There are few records of careful tests of the efficiency of boilers, engines and air compressors of a single plant. In Saunders's "Compressed Air Information," p. 193, a very complete record is given of a 10-hr. test of a plant in operation, supplying air to 14 drills, 14 derrick engines and three small pumps. The test was made by George W. Vreeland and Charles M. Younglove.

The plant is used in excavating the site of the Jerome Park Reservoir, N. Y. (still under construction, 1904), and consists of one Ingersoll-Sergeant Corliss cross-compound condensing air compressor plant, receiving steam from two Hogan boilers, each of 270 H. P. (nominal). The following are some of the data of the test:

Steam gage, lbs. per sq. in.	116.5
Coal used per hour, lbs.	928
Dry steam per hour, lbs.	8,274
Efficiency of boiler	78.1%
Steam per I. H. P., lbs.	17.36
Total I. H. P. of engines	468
Total D. H. P. of engines	402
Mechanical efficiency of engines	86%
Thermal efficiency of compression	82%
Free air per min. per I. H. P., cu. ft.	6
Gage pressure of compressed air, lbs.	67.5
Heat supplied to engine per hr., lb. degs.	9,634,246
Heat utilized by engine per hr., lb. degs.	1,177,437

This is an average size compressor plant, capable of carrying at least 35 drills (3¼-in.); and if it were loaded with 35 drills, each drill would be charged with 265 lbs. of coal, for a 10-hr. run, or 276 lbs. of steam per hour, or with 13.4, I. H. P. of compressor engine. Particular note should be taken of the heat efficiency of the engine, which may be calculated by dividing the 1,177,437 lb. degs. by the 9,634,246 lb. degs., the quotient being 12.3 per cent. Those who have not made a careful study of the compressed air problem would be misled by the statement that the efficiency of compression is 82 per cent., and would be apt to think that this was for the engine and compressor plant. In fact the true heat efficiency of this plant was $12.3 \times .82 = 10.086$ per cent., a trifle more than 10 per cent.—which checks closely with the calculations in the fore part of this chapter.

Gasoline Air Compressors.—Where not more than three or four drills are to be operated, probably no power can equal compressed air generated by gasoline. One pint of gasoline per hour per brake horse power (B. H. P.) of gasoline engine may be counted upon as the average consumption. It will require about 12 H. P. to compress air for each drill (3¼-in. size); hence 12 pints, or 1½ gals., of gasoline will be required per hour per drill while actually drilling. Since gasoline air compressors are self regulating, when the drill is not using air very little gasoline is burned by the gasoline engine driving the compressor. If the drill is actually drilling two-thirds of the working shift, we may safely count upon using about 1 gal. of gasoline per hour of shift per drill, or 8 gals. per shift 8 hrs. long. If gasoline is worth 15 cts. per gal., delivered at the engine, one drill consumes only $1.20 worth of gasoline per shift of 8 hrs. In shaft sinking, tunnel work and the like, as will be shown later, a drill is often idle two-thirds of the shift, so that the gasoline consumption would be still less.

A gasoline compressor possesses other very important economic advantages over a small steam-driven plant. First, there is the saving in wages of firemen; for, once started, a gasoline engine runs itself. Second, there is the saving in hauling or pumping of water and the hauling of fuel. Third, the cost of gasoline is often less than the cost of coal for operating a small plant. The Golden Wave Mine, Congress, Ariz., used a 30 H. P. gasoline driven compressor (Fairbanks-Morse) for sinking an incline, at a cost that was remarkably low. I believe this type of plant is destined to find an increasing field of usefulness. It is especially adapted for rock trench excavation in cities, for tunnel work and for open cuts where only a few drills are operated.

CHAPTER V.

THE COST OF MACHINE DRILLING.

Cost Factors.—The items that go to make up the cost of power drilling are:

1. Wages of driller and helper.
2. Proportionate part of wages of engine and boiler crew.
3. Fuel.
4. Sharpening drills, including transportation to and from shop.
5. Repairs and renewals.
6. Interest and depreciation of plant distributed over the shifts actually worked.
7. Water and oil.
8. Proportionate part of general expenses, such as salaries of superintendent, office employees, rent, etc.
9. Installation of plant, including freight, hauling, setting up, dismantling, etc.

I have enumerated these items because it is a common error to overlook one or more of them; and for the same reason I will now give the factors that determine the number of feet of hole drilled per shift:

1. Character of rock, including resistance to drilling, presence of seams, rapidity of formation of sludge or dust, etc.
2. The percentage of time required to change bits, to shift the drill and to set up.
3. The depth of hole.
4. The size of bits, top and bottom.
5. The use of water jets for removing sludge.
6. The form and sharpness of the bits.
7. The direction of the hole, up, horizontal or down.
8. The percentage of time lost in blasting, breakdowns, sticking of bits, etc.

9. The pressure of the air or steam at the drill.
10. The size of the drill cylinder.

In publishing records of drilling costs, every one of the foregoing factors should be given, yet it is rarely that half of them are recorded. Indeed it is the custom to state merely the kind of rock, the name of the drill, and the cost in cents per foot of hole drilled without any statement as to rates of wages, price of fuel, or in fact any of the data needed to form an intelligent estimate of the applicability of the information to other work.

Percentage of Lost Time.—In operating machines of any kind the percentage of lost time is a factor that should receive the moct careful consideration. Notwithstanding the self-evidence of this fact, I have looked in vain for published records showing the average time lost in setting up, changing bits, cleaning hole, etc. Fearing that my own records of these extremely important items might not cover a sufficiently wide range of conditions, I prepared blanks which I sent to a number of contractors and mine managers. I was not surprised to receive some answers to the effect that conditions were so variable as to make such records of no practical value. Now, it was precisely for the purpose of determining the range of conditions that these blanks were sent out. Indeed, no perfect picture of conditions can be given except by the filling in of just such blanks. They tell at a glance whether the rock was easy to drill or hard; whether the sludge cushioned the blow of the bit or not; whether the drill crew was lazy or not, and, in a word, just what the conditions of operation were, so far as drilling was concerned.

The most serious loss of time in machine drilling is the time lost in changing bits and pumping out the hole; for, with a 2-ft. feed screw (which is the ordinary length), a new drill must be inserted for every 2 ft. of hole drilled. It takes from 4 to 16 minutes, to drill 2 ft. of hole, counting the actual time that the drill is striking, and it ordinarily takes from 2 to 10 minutes to change bits and pump out the hole. I have often timed work where 9 minutes were spent in drilling, followed by 9 minutes lost in changing bits. Counting no other time losses, then, half the available time was lost in the operation of changing bits. From the written reports that I have received, I am certain that few mining men and fewer contractors have ever given this phase of drilling any consideration at all, although, in my judgment, it is often the cause of a loss where there should be a profit. In all the literature on the subject of drilling, I have been unable to find a single mention of the importance of timing drilling operations with the minute hand of the watch.

Where holes are drilled to a depth of 10 ft. or more, the drill steel becomes so heavy that change of bit is an operation usually requiring 3 minutes. When done properly, the driller starts to raise the drill with the feed screw and at the same instant the drill helper begins to loosen the chuck with his wrench. Without any great effort these two operations are finished at the same time, requiring about 1 minute. The pumping out of the hole with the sludge pump can be done by the drill helper in 1 minute, or less, but I have frequently seen deliberate workers take 2 minutes or more. The drill helper then puts a new drill into the hole and enters its shank in the chuck; as soon as this is done the driller should begin to feed the drill forward, and the drill helper should at once tighten the chuck; these operations taking 1 to 1¼ minutes. After the first few blows are struck, it is often necessary to tighten the chuck again, consuming ½ to ¾ minute, but this can be obviated by using good chuck bolts and by training the men properly. The three necessary operations (removing bit, pumping, and putting in new bit) can be done with ease in 3 to 3½ minutes. If, however, the drill helper waits till the driller has raised the drill steel before he begins loosening the chuck, another minute may be unnecessarily added to the time; and, in a similar manner, by deliberation (which is equivalent to laziness) the two men may while away 6 minutes or more unnecessarily. When shallow holes (6 ft. or less), are to be drilled, the drill steel is light and there is often little or no sludge pumping to be done. In such cases it is possible for the driller and his helper to change bits in 1 minute, or even less when they are rushing the work. So far as the changing of bits is concerned the men should be made to work with a vim. When men have to exercise their muscles incessantly for 8 or 10 hours there is reason in taking a slow, steady gait, but in machine work, muscular exercise is intermittent and should be vigorous.

Next in importance to the time lost in changing bits is the time lost in shifting the machine from hole to hole. This, again, is a factor not touched upon by writers, in spite of its importance, especially in drilling shallow holes. To move a tripod from one hole to the next and set up again ready to drill, seldom consumes less than 7 minutes, even when the two men are working rapidly, when the distance to move is short, and when the rock floor is level and soft. When, however, the rock floor is irregular and hard, requiring the vigorous use of gad and pick, not only in making holes for the tripod leg points to rest in, but requiring, also, some little time in squaring up a face for the bit to strike upon, the two men may consume from 30 to 45 minutes, shifting the machine and setting up, if they work deliberately. In such cases it is advisable to have laborers working ahead of the drillers preparing the face of the rock, leveling the site of the hole, removing loose rock, etc. One can see clearly what a great saving in time may thereby be effected; yet, this simple expedient is seldom adopted; but the driller and his helper are usually left to themselves in preparing the ground for each new set up. I repeat again that every foreman and manager of rock excavation should use the minute hand of his watch frequently (and at times when he is not observed), to determine exactly how much time his men are losing in changing bits and shifting. He will thus learn where his employer's money is being wasted.

Where drillers are obliged to move long lengths of air or steam pipe before blasting, or where narrow trenching makes it difficult to shift the drill and causes material delay in advancing the steam pipe line, there are obviously other sources of delay which should be ascertained by careful timing, with a view to reducing the delay by some additional expenditure of money if found advisable.

In seamy or soft rock where the bit sticks frequently in the hole the time lost from this cause alone may be 30 per cent.

Excluding the time required to change bits for the new hole, we may say that two men can ordinarily make a new set up with a tripod in 12 to 15 minutes, if they work rapidly.

When the drill is mounted on a column or bar, the time required to set up the column and get ready to drill ranges from 10 minutes (when the men are racing) up to 60 minutes (when the men are loafing). Men working deliberately ordinarily take about 25 minutes. From one set up of a column, however, 6 to 12 holes may be drilled, by shifting the drill along the column. The shifting itself need not take more than 1 to 2 minutes, but the time required in addition for changing bits and cleaning the hole will not differ materially from the time above given for tripod work.

Tables XVIII and XIX give typical examples of actual work in different kinds of rock and in different parts of the United States. Part of the data was taken from my own notes, but I wish here to express my thanks for most of the data to the following mining and civil engineers: Mr. B. B. Lawrence, Mining Engineer; Mr. E. C. Means, Mining Engineer; Mr. Alex. Veitch, Mining Engineer; Mr. R. Gilman Brown, Mining Engineer; Mr. T. H. Loomis, Civil Engineer; Mr. Walter Seeley, Civil Engineer.

TABLE XVIII.

Average Time Drilling Vertical Holes.
(Drill mounted on a tripod.)

	A	B	C	D	E
Drilling first 2 ft., mins	9	10½	12	8	14
Cranking out and removing bit, mins	1	3½	1	1¼	1½
Cleaning out hole, mins	..	3	1	1¼	1½
Putting in new bit and cranking back, mins	1	2½	1½	1	1½
Drilling second 2 ft., mins	13	10½	..	..	14
Drilling last 2 ft., mins	12	10½	11	6	..
Moving machine from hole to hole and setting up	15	35	..	12	36
Air pressure, lbs. per sq. in	70	(?)	80	70	70
Diameter of drill cylinders, ins	3¼	3¼	3¼	3¼	3¼
Diameter of starting bit, ins	2½	2½	3½	2	2½
Diameter of finishing bit, ins	1¾	1½	1¼	1¼	2
Depth of hole, ft	12	6	20	12	6
Kind of rock	Lm.	S.	Gr.	Sd.	Tr.
Length of shift, hrs	10	10	10	10	10
Ft. drilled per shift	..	..	48	96	36

Note.—The kind of rock designated by the abbreviations is as follows: Lm., limestone; S., sandstone (hard); Gr., granite; Sd., sandstone (soft); Tr., trap (diabase).

TABLE XIX.

Average Time Drilling Holes in a Breast.
(Drill mounted on a column.)

	F	G	H	I	J	K	L
Drilling first 2 ft., mins	20	10	5	10	5¼	3 1-3	24
Cranking out and removing bit, mins	2	3	¾	3	½	1-3	2
Cleaning out hole, mins	2	3	0	3	0	0	0
Putting in new bit and cranking back, mins.	1	2½	1	4	¼	1½	2
Drilling second 2 ft., mins	15	9	6	10	19	4¾	30
Drilling last 2 ft., mins	15	9	8	10	..	..	30
Shifting machine on column, hole to hole, mins.	5	7	8	3½	5	5	10
Shifting column, and setting up, mins.	20 to 60	25	25	18	31	30	40
Air pressure, lbs. per sq. in	75	75	80	100	75	75	70
Diameter of drill cylinder	3¼	3¼	3¼	3⅛	2	2½	3½
" " starting bit	2¾	2½	2½	2	1⅜	1¾	2¼
" " fiinishing bit	2	1½	1¾	1½	1	1¼	1½
Depth of hole	8	12	12	6	4.6	5.5	5.5
Kind of rock	Sp.	Sd.	Lm.	Gr.	Py.	Py.	Sl.
No. of holes drilled at one column set up	9	10	12	10	7	8	7
Length of shift, hrs	10	10	10	8	8	8	10
Time lost at blasting, hrs	1	1½	(?)	½			(?)
" " mucking and timbering, hrs.	4	(?)	(?)	..	2	1	(?)
Ft. drilled per shift	..	..	..	60	31	38	25

Note.—The kind of rock designated by the abbreviations is as follows: Sp., soapstone; Sd., sandstone; Lm., limestone; Gr., granite; Py., quartz porphyry (soft in column J; hard in K); Sl., slate. Column J applies to Rand drills using chisel bits; column K to Leyner-Water drills using X bits in a softer rock.

Results of a Drilling Contest.—The following is an abstract from an article in the *Mining Reporter,* July 17, 1902, reprinted in the Leyner drill catalogue. Eleven Water-Leyner drills (model 5), seven Ingersoll drills and five Sullivan drills (all 3-in.) were entered in a drilling contest at Idaho Springs, Col. Each drill was run by an experienced driller and helper, the different contestants bringing their own drills. A face, or breast, was prepared near the Newhouse Tunnel, where the rock is a "schist of more than average hardness for drill work." Each drill was to put in two 9-ft. holes, one looking up, one down, the angle not exceeding 25° from the horizontal. The finishing bit was 1⅜ ins. diam. The air pressure was 110 lbs. I have summarized the results of this contest as follows: Nine of the Leyner drills finished the contest, three of the Ingersoll drills and two of the Sullivan drills. Table XX. gives the average results of the drills that finished, as well as the best results of an individual drill of each of the three makes.

TABLE XX.

Make of Drill.	Setting Up.		Drilling 1st Hole.		Moving 1st to 2d Hole.		Drilling 2d Hole.		Tearing Down.		Total Time.	
	Mins.	Secs.	Mins.	Secs.	Mins.	Secs.	Mins.	Secs.	Mins.	Secs.	Mins.	Secs.
Leyner (average)...	4	10	28	20	1	00	23	40	3	10	60	20
Ingersoll (average)	4	35	24	15	1	20	34	30	4	15	69	05
Sullivan (average).	3	30	17	30	0	30	17	50	2	55	42	15
Leyner (best)......	5	05	26	40	1	0	42	15	3	15	78	15
Ingersoll* (best)...	4	30	27	20	1	25	35	50	4	30	73	35
Sullivan (best).....	5	0	23	0	0	30	22	0	3	10	53	40

In studying this table it is well to note that the first hole was the up hole, and the second hole was the down hole in all cases. It will be noticed that the down hole required much longer to drill than the up hole in the cases of the Inger-

* This drill was disqualified because the second hole was 4⅜ ins. short.

soll and the Sullivan drills. The Leyner drill, on the contrary, showed very little difference in speed of drilling either up or down holes; the reason being, I think, due to the fact that the water (under pressure) that is used with the Leyner drill keeps the bottom of the hole clean in all cases (up or down holes), leaving no cushion of sludge for the bit to strike upon. The Ingersoll and the Sullivan drills worked to better advantage in the up, or dry holes, than in the down, or wet holes; because the water used in the wet holes was not forced in through a pipe, but merely thrown into the hole with a tin cup (as is the ordinary method of "tending chuck"), and in a rock that makes sludge rapidly, as many schists do, the sludge accumulates under the bit and cushions its blow. In an up (dry) hole, however, the chips and dust roll out of the hole as fast as formed, so that much better speed is possible in a rock that cuts rapidly, as this particular "schist" evidently does. The Leyner drill is an excellent drill in soft rocks, and drills well also in hard rocks, but with a greater air consumption than any other make of drill, as is shown on page 55.

Rule for Estimating Feet Drilled per Shift.—We are now possessed of sufficient data to enable us to formulate a rule whereby the number of feet drilled per shift, under given conditions, may be predicted. I will not go into the method that I used in deducing the following rule, which is strictly correct, for the method is one of simple arithmetic. The rule is:

To find the number of feet of hole drilled per shift divide the total number of working minutes in the shift by the sum of the following quantities: The number of minutes of actual drilling required to drill one foot of hole, plus the average number of minutes required to change bits divided by the length of the feed screw in feet, plus the average number of minutes required to shift the machine from hole to hole divided by the depth of the hole in feet.

Suppose, for example, the shift is 10 hrs. long, that is

600 mins.; that it requires 5 mins. to drill 1 ft. of the rock; that it requires 4 mins. to change bits and clean hole; that the feed screw is 2 ft. long; that the machine can be shifted from hole to hole in 16 mins.; and that each hole is 8 ft. deep. Then according to the rule we have: The number of feet of hole per shift is $600 \div 5 + \frac{4}{2} + \frac{16}{8}$, which is equivalent to $600 \div 9$, or 66 2-3 ft. drilled per 10-hr. shift.

For those who can use simple algebraic formulas the above rule is much more compactly expressed in the following formula:

$$N = \frac{S}{r + \frac{m}{f} + \frac{s}{D}}$$

N = number of feet drilled per shift.

S = length of working time of shift in minutes = 600 for a 10-hr. shift when no time is lost by blasts, breakdowns, etc.

r = number of minutes of actual drilling required to drill 1 ft. of the rock.

m = number of minutes required to crank up, change drills, pump out hole and crank down.

m = 3 to 4 mins. ordinarily.

f = length of feed screw, in ft., ranging from 1¼ ft. in "baby" drills to 2½ ft. in largest drills, but ordinarily 2 ft.

s ~~S~~ = number of minutes required to shift machine from one hole to the next, including the time of chipping and starting the new hole, but not including the time of cranking up and cranking down.

s ~~S~~ = 5 mins. for very rapid shifting of a tripod machine on level rock.

s ~~S~~ = 12 mins. for moderate speed of shifting a tripod machine on level rock.

s ~~S~~ = 20 mins. for very deliberate shifting of tripod machine on level rock.

s ~~S~~ = 30 to 40 mins. for difficult set up of tripod in irregular rock surface.

$S = 25$ mins. divided by the number of holes drilled from one column set up (when columns are used) plus 2 mins.

$D =$ depth of hole in ft.

Even a casual study of the foregoing formula, or rule, must impress the practical man with the importance of the lost time elements in machine drilling; consequently of the value of timing the operations of changing bits and moving machines when the men do not know that they are being timed. Another feature that stands out strikingly is the reduced output of a drill working in a shallow hole. Let the reader solve a few problems, assuming first an average depth of hole of 16 ft. and finally an average depth of only 2 ft. (such as occurs often in the skimming work in road building), and he will never make the blunder of the contractor who bid the same price for rock excavation on the 2-ft. deepening of the Erie Canal as had been bid for the 36-ft. excavation on the Chicago Canal.

The best way of showing the remarkable effect that the depth of hole has upon the number of feet drilled, when the drill is mounted upon a tripod, is to apply the rule given on page 79. If we assume that the shift is 10 hrs. long; that the rate of drilling is 1 ft. in 5 mins.; that it takes 4 mins. to change bits and pump out the hole at each change of bits; that the feed screw is 2 ft. long; and that it takes 15 mins. to shift from one hole to the next; by applying the rule we obtain the following results:

Depth of hole, ft.	1	2	3	5	10	15	20
Feet drilled in 10 hrs.	27	41	50	60	70	75	80

When drillers are lazy they may readily consume 8 mins. in changing bits and pumping out the hole each time. With all conditions the same as before, excepting that 8 mins. are consumed in changing bits, we have the following results:

Depth of hole, ft.	1	2	3	5	10	15	20
Feet drilled in 10 hrs.	25	36	43	50	57	60	62

It will be seen that in deep hole drilling 20 per cent. decreased efficiency results from just a little laziness in changing bits, under the conditions assumed; and in softer rocks the percentage of decreased efficiency is much greater. Where the holes are shallow the time involved in shifting from one hole to the next becomes an important factor. Assuming that the conditions are the same as in the first instance, except that 30 mins. are consumed in shifting from one hole to the next, then we have the following results:

Depth of hole, ft.	1	2	3	5	10	15	20
Feet drilled in 10 hrs.	16	27	35	46	60	67	70

In similar manner I might tabulate other results derived by varying the different time elements in drilling; but enough has been given to show the supreme practical importance of studying these details, which so many practical men have apparently ignored. I leave it to the reader to apply the rule, or formula, to other cases, for the results of such personal application of the rule will stick in the memory and be of more real value than much reading of tabulated information.

Rates of Drilling in Different Rocks.—Unfortunately no published record exists showing rates of drilling in different kinds of rock with given air or steam pressures and given sizes of drill bits. Such scattering records as are to be found merely give the feet of hole drilled per shift. Tables XVIII. and XIX. give a fair idea of the speed of actual drilling, and from them together with other data obtained by observation I have compiled the following table for drilling with 3⅛-in. machines using air or steam at 70 lbs. pressure, starting bit about 2¾ ins. and finishing bit about 1½ ins.:

	Time to drill 1 ft.
Soft sandstones, limestones, etc........	3 mins.
Medium, ditto	4 "
Hard granites, hard sandstones, etc....	5 "
Very hard traps, granites, etc.	6 to 8 "
Soft rocks that sludge rapidly.......	8 to 10 "

The foregoing data apply only to drilling where no time is lost by the sticking of the bit in the hole, and only with air pressure and bits approximately as above given. The reader should now read again the text on pages 52 to 55, noting especially the falling off in the rate of drilling accompanying decreased air pressure. He should also study the table on page 54, relating to the effect of size of bit upon speed of drilling, remembering that all the tests there recorded relate only to shallow holes. As holes grow deeper the bit grows smaller, but at the same time the drill steel grows heavier, in consequence of which the last 2 ft. of a deep hole, with a bit only 1½ ins. in diam., are ordinarily drilled no faster than the first 2 ft. with a much larger bit. With a powerful drill and a water jet it is quite possible that the last 2 ft. might be drilled faster than the first 2 ft.

Note especially that if a water jet is not used, drilling may actually be slower in a soft, friable rock, like shale, than in the toughest trap. This fact is well brought out in Table XIX., column "K," where the drilling of the second 2 ft. of hole consumed 19 mins.! Yet this material was a soft porphyry that with a water jet was penetrated at the rate of 2 ft. in less than 5 mins., as shown in column "K." While the Leyner-Water drill is an excellent machine for drilling shallow holes in rock that makes sludge rapidly, its excellence is due primarily to the use of water under pressure. In drilling granite its speed is no greater than that of the best types of the ordinary percussion rock drill.

Reverting to the subject of air or steam pressure as affecting the speed of drilling, it is not always easy to gage the pressure at the drill; but I have found it a simple matter to estimate the pressure with considerable accuracy by noting the number of blows per minute struck by the drill. Using drill-steel of given weight, first test the drill for speed at a point so near the compressor or boiler that little or no loss of pressure can occur in the pipe line. Vary the pressure from, say, 80 down to 40 lbs. in making the test of the number of blows struck per minute, and thereafter your pencil and your watch will be a good enough pressure gage. The number of blows can be counted by keeping time with a pencil. The pencil, in the hands of the observer, is made to strike a sheet of paper every time the drill strikes the rock. Then at the end of a definite number of seconds, say 15 secs., the number of marks of the pencil point upon the paper are counted, and the number of blows per minute are computed.

Once it has been ascertained how many blows a given size drill strikes per minute when working at full stroke, under varying gage pressures, the only gage needed is the pencil, and a watch; for the pressure can be roughly ascertained by determining the number of blows struck per minute. This I have found to be an exceedingly useful means of ascertaining whether or not too many drills are drawing power from a given pipe line. If a given steam boiler or compressor is designed to supply 15 drills, and if it is afterward loaded up with 20 drills, the pressure at each drill will be reduced, resulting in a very decided falling off in the number of feet of hole put down daily by each drill.

Average Footage Drilled per Shift.—In subsequent chapters many records of actual work are given, but that the inexperienced reader may have a good general conception of what constitutes a day's work under ordinary conditions the following summary may be of benefit: In drilling vertical holes, with the drill on a tripod, the holes being from 10 to 20 ft. deep, shift 10 hrs. long, I have found that in the hard "granite" of the Adirondack Mts., N. Y., 48 ft. is a fair 10-hr. day's work. In the granites of Maine and Massachusetts 45 to 50 ft. is a day's work. In New York City, where the rock is mica schist, deep holes are drilled at the rate of 60 to 70 ft. per 10-hr. shift by men willing to work, but 40 to 50 is nearer the average of union drillers. In the very hard trap rock of the Hudson River 40 ft. is considered a fair day's work. In the soft red sandstone of northern

New Jersey 90 ft. are readily drilled per day wherever the rock is not so seamy as to cause lost time by the sticking of the bit; in fact, I have records showing 110 ft. per 10-hr. shift in this rock. In the hard limestone near Rochester my records show about 70 ft. per 10-hr. shift. In the limestone on the Chicago Drainage Canal 70 to 80 ft. was a 10-hr. day's work. In the hard syenite of Douglass Island, in open pit work, and where it is difficult to make set-ups, 36 ft. is now the average per 10-hr. day. In the limestone near Windmill Pt., Ontario, 3⅝-in. drills average 75 ft. a day (holes 18 ft. deep); 2¾-in. drills, 60 ft. a day, and "baby" drills, 37 ft. a day.

The foregoing examples all apply to comparatively deep vertical holes, in open excavation. In tunnel work there is no reason why a drill should not do about the same work per shift, were there no delays in timbering, mucking, waiting for gases to clear, etc. Such delays, however, often reduce the drill footage very much, as will be seen in subsequent chapters.

Cost of Sharpening Bits.—In the South African gold mines each machine averages 20 drills per shift, and taking the average weight at 15 lbs. per drill, it is evident that 600 lbs. of drills must be carried to and from the shop for each machine per shift. Where 30 drills are at work this means the transportation of 9 tons of drills each shift—an item in itself worthy of consideration.

One blacksmith (with a helper) will sharpen about 140 bits a day, and under ordinary conditions will keep 5 to 7 drills supplied with sharp bits. In average rock a bit must be sharpened for every 2 ft. of hole; in very soft rock a bit for every 4 ft., and in very hard rock a bit for every 1½ ft. of hole.

Sharpening 140 bits by hand costs $3 for blacksmith's wages, $2 for helper and 60 cts. for charcoal, or 4 cts. per bit.

Mr. Edward D. Self, Manager and Engineer San Carlos

Copper Co., San José, Mexico, writes me that 185,828 hand drills were sharpened in drilling about 65,000 ft. of hole, at a cost of 2 cts. per bit (Mexican currency); and that 10,000 machine drill bits were sharpened by hand at a cost of 3 cts. each (Mexican currency).

For sharpening large numbers of bits daily, drill sharpening machines are much cheaper than hand sharpening, as shown on page 30.

Cost of Drill Repairs.—Mr. Thomas Dennis, agent of the Adventure Consolidated Copper Co., Hancock, Mich., has kindly furnished the following data of the average monthly cost of keeping a drill in repair:

Supplies for repairs	$1.31
Machinist labor	8.45
Blacksmith labor	1.60
Total repair charge per month	$11.36

The number of drills in the shop at any one time is about 15 per cent. of the total number. This low cost is based upon work where a large number of drills are used and well handled by the users.

I am indebted to Mr. Josiah Bond, mining engineer, for the statement that the cost of repairs averages 50 cts. per drill per shift in mines where a few drills are operated and renewal parts purchased from the manufacturers. In open cut work my experience is that 75 cts. per drill per shift is a fair allowance for renewals and repairs. In the gold mines of South Africa, where each drill works two shifts per day, the cost of drill repairs is $300 per drill per year; while the first cost of a 3¼-in. drill with bar is $185, according to a recent report of the Government Mine Inspector.

Plant Rental.—This term is a convenient one to use instead of "interest and depreciation," moreover "plant rental" is a term that clearly excludes the cost of current repairs. It is exceedingly difficult to name any definite percentage to be allowed for rental of a drilling plant, because the term of years during which the plant will be operated by the purchaser is usually a matter of guess work. Little or nothing can be borrowed from banks upon a drilling plant as the only collateral; hence the owner of a plant is not justified in charging only the legal rate of interest for rental even if the plant were to have an everlasting life. The owner should charge for interest on the plant all that he could earn in the way of profits if he had the plant money invested otherwise in his business. The life of a plant is only a few years at best, since improvements in machinery are constantly taking place. Moreover "dull times" put scores of plants upon the idle list for indefinite periods of time. In view of these facts it is good business policy to charge the whole cost of a drilling plant up against the contract job for which it is purchased, if the job is a large one. For small mines and small jobs generally, it is wise to charge up 20 to 40 per cent. of the first cost of the plant for annual "rental" or "sinking fund." Divide this annual "rental" by the total number of shifts that will probably be worked during the year, to arrive at a probable rental cost per shift; and in estimating the probable working days in the year make liberal allowance for strikes, bad weather and sundry delays. In the estimates of cost that follow I purposely omit an allowance for "plant rental," because each case must be treated as a problem in itself; nevertheless, "plant rental" is often a very important item and one that is not infrequently forgotten entirely in making estimates of cost.

Cost of Installing a Compressor Plant.—The following is an itemized account of the cost of installing a small compressor plant. The compressor was a Rand, Class C, 24 x 30-in., that cost $4,000. The boiler was a second-hand 150 H.-P. locomotive boiler that cost $1,000. This plant was capable of furnishing 1,300 cu. ft. of free air per min. at 80 lbs. pressure, or enough to run 10 or 12 drills. Cost of installing boiler:

22 days	laborers, at $1.50	$33
23 "	engineers, at $3	69
13 "	mechanics, at $4	52.
13 "	" help, at $2	26
1 "	bricklayer, at $4	4
	Total	$184

Cost of installing compressor:

120 days	laborers, at $1.50	$180
4 "	engineers, at $3	12
22 "	mechanics, at $4	88
80 "	" help, at $2	160
50 "	carpenters, at $3	150
3 "	bricklayers, at $4	12
6 "	teams, at $4	24
8 "	foremen, at $3	24
	Total	$650

Cost of materials:

15M lumber for housing compressor, at $25	$375
1,400 sq. ft. tar paper (1 layer)	21
32 cu. yds. concrete, at $4	128
5M brick, at $7	35
6 bbls. cement, at $2	12
Sand	1
Total	$572

Cost of a Large Compressor Plant.—The following is the estimated cost of a compressed air plant for a western mine, the compressor being designed to carry 20 drills (3¼-in. size):

4 high pressure boilers (66-in. x 16 ft)	$6,000
Housing and installing boilers	2,000

Duplex compound air compressor	16,000
Housing and installing compressor	2,000
Pipe, 1,000 ft. of 6-in., and 1,500 ft. of 1 in.	1,200
Machine shop and tools	800
Total	$28,000

It is usually safe to estimate on a basis of $130 to $150 per drill for the cost of a large and efficient compressor plant and pipe line, to which must be added the cost of the drill itself, which is about $250 for a 3¼-in. drill mounted.

Cost of Operating Drills.—When operating a single (3¼-in.) drill supplied by steam from a small portable boiler, I find the cost is usually as follows for a 10-hr. shift:

1 drill runner	$3.00
1 drill helper	1.75
1 fireman	2.00
660 lbs. of coal (0.3 ton at $3)	.90
Water if hauled, say	.75
Hauling and sharpening 30 bits (incl. new steel) at 4 cts.	1.20
Repairs to drill and hose renewals	.75
Total per 10 hrs.	$10.35

The foregoing is merely an example, based, however, upon several different jobs; but in each case the accessibility of a blacksmith, the nearness to water, the price of coal delivered at the boiler, etc., must be determined before an accurate estimate can be made. If 4 drills, for example, are to be operated from the same boiler, the fuel bill will be somewhat reduced even if the pipes are not covered with asbestos, and of course the wages of the fireman will be distributed over 4 drills. It will then pay to have a blacksmith at hand. If 10 or more drills are run by steam from a central boiler, and if the main pipes are lagged, the fuel should not much exceed 300 lbs. per drill per 10-hr. shift. By the rules previously given a fairly close estimate can be made of the number of feet of hole that each drill should average. If 60 ft., for example, is to be a fair day's work in limestone or sandstone, we have $10.35 ÷ 60 = 17 cts. per ft. as the cost, exclusive of superintendence, plant installation and plant rental.

If a central compressor or steam plant supplies power for, say, 15 drills, we may estimate the cost of operating each drill as follows:

1 drill runner	$3.00
1 drill helper	1.75
1-15 fireman at $2.25	.15
1-15 compressor man at $3	.20
300 lbs. coal (water nominal) at $3 ton	.45
Sharpening bits, 30 at 3 cts.	.90
Repairs to drill, hose, etc.	.75
Total for 60 ft. of hole at 12 cts.	$7.20

If the cost of each drill and 1-15 part of the compressor plant is $350, and 30 per cent. of this is assumed as a fair allowance for annual plant rental, we have $105 to charge up against each drill for "rental," or about 50 cts. per shift if 200 shifts are worked each year, or about 1 ct. per ft. of hole drilled.

Cost of Drilling Blast Holes with a Well Driller.—In *Engineering News* I first described the use of well drillers on the Pennsylvania Railroad work for drilling blasting holes. The well drillers used were the ordinary type of portable driller, consisting of a wagon on which is mounted a 4 to 8 H.-P. engine that drives a walking beam; the walking beam raises and lowers a rope, to which is fastened the churn bit and rods that form the "business end" of the driller. A 5⅝-in. bit was used in this work, and even with this large bit each drill averaged three 20-ft. holes, or 60 ft., drilled in shale per 10-hr. shift. In limestone, however, and in hard sandstone not more than 10 ft. of hole were drilled per shift. In describing the use of the well driller for this purpose I suggested that if the bits were reduced to about

3 ins. in diam., and if the drill rods were suitably weighted, much better progress would be made in hard rock. Shortly afterward the Cyclone Drilling Machine Co., of Orrville, Ohio, put upon the market a driller especially designed for contractors doing railroad work. The Cyclone driller has a 5 H.-P. engine, uses a 3-in. bit, has drill rods that screw together and a suitable weight to give power to the blow, and the whole outfit on trucks weighs only 5,000 lbs. Fig. 6 shows one of these drillers at work on the Wabash Rail-

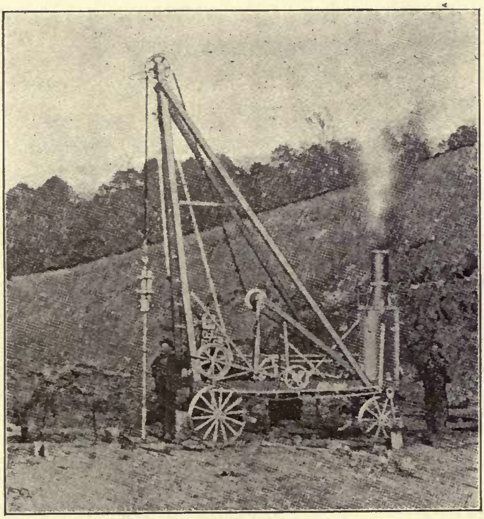


Fig. 6.

road. For this type of driller I predict an increasingly large field, for the following reasons: (1) A drill will not stick in the hole, because of the powerful direct pull of the rope that operates the drill rods; (2) there is no limit to the depth of the hole, and the deeper it is (up to any limits possible in blasting) the better the drill works, due to the increased weight of the rods; (3) this type of drill consumes less fuel than the ordinary steam drill; (4) the weight of bits to be carried back and forth from blacksmith shop is much less than for the ordinary machine drills; (5) the driller will drill through the earth overlying the rock, so that no stripping is necessary. Mr. W. M. Douglass, of Douglass Bros., contractors, was kind enough to keep records for me showing the cost of operating one of these cyclone drillers compared with a Rand drill. The following are the data:

The holes were drilled with bits to give 3 ins. diam. at the bottom of the hole. Holes were 24 ft. deep in solid brown sandstone in eastern Ohio. In 14 days, of 10 hrs. each, the driller put down 692 ft., or practically 50 ft. per day. The daily cost of operating was as follows:

Drill runner	$3.00
Drill helper and fireman	2.00
Pumping water	.60
6 bu. (480 lbs.) coal at 10 cts.	.60
Total for 50 ft. of hole	$6.20

This gives a cost of 12½ cts. per ft. of hole, not including interest and depreciation and bit sharpening. The best day's work in the brown sandstone, using all the weights, was 53 ft., but in blue sandstone, which was softer, 60 ft. were drilled per day using light weights.

In the same brown sandstone cut an 8-day test was made with a 3¼-in. Rand drill for comparison. The holes were 20 ft. deep, 1¾ ins. diam. at the bottom (as against 3 ins. with the well driller), and 28 holes were drilled in the 8 days, making 70 ft. the average day's work. A 10 H.-P. boiler furnished steam. The daily cost of operating the Rand drill was:

Drill runner	$3.00
Drill helper	1.50
Fireman	2.00
Water	.75
10 bu. (800 lbs.) coal at 10 cts.	1.00
Total for 70 ft. of hole	$8.25

This was equivalent to 11.8 cts. per ft. of hole, not in cluding interest and depreciation and drill sharpening.

It should be observed that the well driller holes were deeper (and they could have been still deeper without increasing the cost per foot) as well as larger in diameter than the Rand drill holes. The greater diameter saved a considerable amount of dynamite in springing the holes, since each well driller hole was sprung three times, as compared with four or five times for the Rand drill holes, in order to make a chamber large enough to hold the black powder. Mr. Douglass has made some interesting tests on the use of black powder and dynamite in alternate rows of holes, for which see page 149.

I would suggest a further improvement in this type of driller, namely, the use of a gasoline engine instead of a steam engine. Such a change would do away with the cost of water for the boiler. It would also make it unnecessary to have a fireman.

Cost of Drilling with Electric Drills.—Upon this subject there is practicallly nothing in print. I am indebted to Mr. J. B. Hobson, Manager Caribou Hydraulic Co., Bullion, B. C., for the following cost data: Four Gardner Electric Drill Co.'s No. 15 drills, with 2 H.-P. motors, and one "B" drill with a 1½ H.-P. motor, have been used for two years by Mr. Hobson with excellent results. Each of the larger drills has averaged 13 holes, 8 ft. deep, in firm augite diorite and porphyrite, per 10-hr. shift. The starting bit is 2½ ins. and the finishing bit 1½ ins. in diameter. The cost, per 10-hr. shift, of operating three drills has been as follows:

1 cord of wood	$2.25
1 electrical engineer	4.00
3 drillers, at $4	12.00
3 helpers, at $2	6.00
1 blacksmith	4.00
1 blacksmith's helper	2.00

3 bu. charcoal	$0.75
Oil	.55
Total for 3 drills, 312 ft. drilled	$31.55

The cost of drilling, including sharpening, but excluding interest and depreciation, was 10 cts. per foot of hole. See page 20 for cost of hand drilling in the same rock.

In *Trans. Am. Inst. Min. Eng.*, 1903, Mr. Frank E. Sheppard describes the box electric drill which is manufactured by the Denver Engineering Works, Denver, Colo. The drill steel is not churned but is struck by the piston, resembling the Leyner drill in this respect. Water is forced down through a pipe which slips over the drill steel, and the sludge is washed out of the hole. In drilling a block of granite from Platte Canyon, Colo., an electric current of 11 amperes at 110 volts was used, which is equivalent to 1.62 H. P. A hole 2¼ ins. in diameter was drilled, the first 16 ins. being drilled in 7 mins.; then 1 min. was consumed in changing bits. The next 14 ins. were drilled in 5 mins., and 1 min. more was consumed in changing bits. The last 3½ ins. were drilled in ¾ min. The rate for the full 33½ ins. was 2.23 ins. per minute. In tunnel work in Boulder County, Colo., one machine drilled 12½ ft. of holes (number of holes not given), 2¼ ins. diam., in hard granite in 2 hrs. Three tunnel holes, each 5 ft. deep, were drilled in 2½ hrs. In shaft sinking at the Ophir Mine, Anaconda, Colo., five 4-ft. holes were drilled in 3½ hrs. In the tunnel of Imogen Basin Gold Mines Co., Ouray, Colo., four holes 5 ft. deep, were drilled in 3 hrs. in soft rock with mud slips running through the seams.

CHAPTER VI.

COST OF DIAMOND DRILLING.

For determining the nature of bridge foundations the character of proposed canal or railway excavations and for prospecting for mineral deposits, the diamond drill is an invaluable machine. The bit of a diamond drill consists of a number of diamonds mounted on the end of a hollow tube. This bit is rotated by hand, steam, air or electric power, while at the same time water is pumped down the hollow drill rods and passes up outside of the rods, carrying away the rock dust made by the grinding of the diamonds against the rock. The bit cuts an annular channel, leaving a core of rock inside the core barrel. When the drill has penetrated the rock a distance of 6 to 10 ft., the drill rods are raised and the act of raising them breaks off the rock core, which is brought to the surface in the core barrel and kept for examination.

The diamonds are preferably black diamonds, known in the trade as "carbons"; but where the rock is soft, white diamonds, known as "borts," may be used. Sometimes both kinds are used in one bit. A bit usually has 6 to 8 carbons weighing 1 to 1½ carats each. Small stones are not economical because after a carbon has been worn down so that it weighs less than about ½ carat it cannot be reset. In selecting carbons reject those showing a cokey structure, also those having thin, sharp edges. Carbons having straight edges with sides forming an obtuse angle of 95° to 140° are most durable. The cleavage should be tested with a pair of hand pincers. Old stones that have been used are to be preferred since a poor stone will break in use, and no test is so satisfactory as the test of usage. The carbons selected for a bit should be quite uniform in size.

When diamond drilling was first introduced into this country it was predicted that it would be used exclusively for drilling blast holes, and in fact diamond drills were used on the Sutro tunnel for a while, and in sinking one or two shafts by the "long hole" method, which involved drilling holes several hundred feet deep, filling them with sand, then removing the sand for about 8 ft., charging with powder, firing, and so on. The development of machine drills using steel bits and the steady rise in price of carbons have together shown these early predictions to have been fathered by hope rather than by reason.

The sizes of holes and cores are as follows:

Hole, diam. in ins....	1⅜	1¾	2	2⅝	3 9/16
Core, " " "	15/16	1 3/16	1 7/16	2	2⅞

Price of Diamonds.—In 1873 the price of carbons per carat was $8 to $12, whereas now the price is about $50. I am indebted to the Standard Diamond Drill Co., of Chicago, and to the Yawger-Lexow Co., of New York, for the following statements as to the average cost of carbons per carat from 1895 to 1903:

1895.	1896.	1897.	1898.	1899.	1900.	1901.	1902.
......	$36	$50	$60	$55	$50	$45	$50
$18.50	$28	$35.50	$35.50	$36	$51.50	$48.50	$47

It will be noted that these firms do not agree very closely as to prices prior to the year 1900. The American Diamond Rock Drill Co., of New York, quoted $52 per carat for best selected carbons and $16 per carat for best selected borts in November, 1902.

There is no import duty on carbons in the United States, Canada o1 Mexico.

Water Required.—In boring a 2-in. hole where the progress is about 10 ft. per 10-hr. shift, from 100 to 125 gals. of water are required to wash out the sludge formed in drilling, provided the water is used but once. In cases where the water is expensive it is customary to collect the return water in a settling tank and use it over and over; and, unless a large amount of water escapes through crevices, 30 or 40 gals. per shift will be consumed by evaporation and leakage.

Price of Diamond Drills.—A hand power drill that can be used to bore a 1⅜-in. hole (giving a 15-16-in. core) up to a depth of 350 ft.; or a 2⅝-in. hole (giving a 2-in. core) up to a depth of 250 ft., will cost approximately $850 f. o. b. New York or Chicago. This includes 300 ft. of pipe, 6 carats of carbons, all tools, etc., necessary. The machine alone weighs 330 lbs., and can be divided into packages weighing 40 lbs.; but the whole outfit packed for shipment weighs 2,800 lbs. If it is desired to run this drill by horse power, $60 additional will purchase the horse power equipment. A hand power plant capable of drilling 50 per cent. deeper than the above costs about $1,400.

A steam power plant that can be used to bore a 1⅜-in. hole 800 ft. deep, or a 2⅝-in. hole (2-in. core) 500 ft. deep, costs about $2,400, including the 8 H. P. boiler on wheels; the drill itself costing $1,100, the boiler $400; 1 set of carbons (9 carats), $450, and the balance for sundries. The drill itself weighs 600 lbs., but the full outfit packed for shipping weighs 10,000 lbs.

A steam power plant that can be used to bore a 1¾-in. hole 1,500 feet, or a 2⅝-in. hole 1,000 feet, can be purchased for $4,600; of which $2,400 is for the drill, $500 for the 15 H. P. boiler on wheels, $600 for 12 carats of carbons and the balance for rods and sundries. This outfit weighs 20,000 lbs.

Cost in Virginia.—There is a great deal to be found in print relative to the cost of diamond drilling, but unfortunately the records as published are in such form as to be of far less value than they should be. By this I mean that any record of any kind of drilling to be of great value should give: (1) The rate of penetrating a given kind of rock when the drill is actually cutting; (2) the speed, power and weight of the machine; (3) the time lost in raising the drill to change bits, remove cores, or the like; (4) the time required to shift from one hole to the next; (5) the average time lost in repairs, breakdowns, etc.; (6) the diameter and depth of hole; (7) the time consumed in driving and pulling casing. No record in print contains all these factors. Strangely enough, one of the earliest printed accounts contains more of these factors than any subsequent record. I refer to an admirable paper by O. J. Heinrich, in *Trans. Am. Inst. Min., Eng.*, 1874, from which I have abstracted the following:

The diamond drill crew consisted of three men, two to run the drill and one to help raise the drill rods, beside a foreman. The shift was 12 hrs. long, and the following was the cost of operating a shift:

Foreman, or boring master	$2.50
Mechanic, or engineer	2.00
Assistant	1.50
Laborer	1.00
Total labor	$7.00

The coal consumed was 10 lbs. per H. P. per hr. For holes up to 1,000 ft. deep an 8 H. P. engine was used, the drill rods weighing 4,500 lbs.; but up to a 1,500-ft. hole a 12 H. P. engine was used, with rods weighing 7,000 lbs. The drill had a 2-in. bit, on which were mounted never less than 12 carbons, better 16. The drill rods were raised after every 10 ft. of drilling. The drilling was done in Chesterfield county, Va., prospecting for coal, in 1873. The cost of operating per shift is given as follows:

Labor	$6.50
1/3 ton coal at $3	1.00
Oil	0.50
Diamonds and repairs	11.00
Interest and deprec.	1.92
Total per day	$20.92

The price of carbons was $10 per kt. Rates of wages were also much lower then, and it should be noted that the allowance for interest and depreciation is too low for a plant costing $7,200, as it is stated this 8 H. P. plant cost.

Depth of hole in earth and rock..........	419	850	1,142
Depth bored in rock......................	396	826	1,118
No. of 12-hr. shifts actually boring.....	13.88	44.41	59.29
" " " " raising rods	15.87	59.34	116.46
" " " " incidentals.........	3.25	15.25	68.25
" " " " total.................	33.00	119.00	224.00
Ft. progress per hr. while boring........	2.37	1.55	1.57*
" " " " average	0.998	0.578	0.308
Cost of labor, per ft..................	$0.36	$0.59	$1.02
Cost of fuel ($3 ton) per ft............	$0.53	$0.14	$0.17
Cost of all other items, incl. materials and blacksmithing	$1.29	$1.43	$2.05
Interest	$0.16	$0.27	$0.38
Total cost per ft......................	$1.86	$2.43	$3.62

From the data given by Mr. Heinrich I have prepared the following formulas to be used in computing the number of hours required to drill a hole of given depth.

Let

T = Total number of minutes required to bore the hole.

n = total depth of hole in feet.

l = length of each coupling rod = 10 ft. in this case.

t = the number of minutes required to bore 1 ft. of the hole. In the formation given by Heinrich t = 19 mins. per ft. of hole up to a depth of 300 ft., to which add 5 mins. per ft. for each 100 ft. of increased depth.

r = time in minutes required to raise and lower the rods, including 2 mins. to uncouple and couple up.

r = 7 mins. for hole up to 300 ft. plus 1-3 min. for each additional 100 ft.

s = number of lengths of coupling rod.

The time consumed in actual boring in feet is obviously nt. The time consumed in raising and lowering the drill rods is the sum of an arithmetical series in which s = the number of terms and r = the common difference; hence the sum is $\frac{1}{2}s\,(2r+[s-1]\,)r$, which reduces to $\frac{s(1+s)r.}{2}$ The total time is therefore:

$$T = nt + s \frac{(1 + s)}{2} r$$

$$s = \frac{n}{l}$$

$$T = nt + \frac{n\,(l + n)}{2\,l^2} r$$

If $l = 10$

$$T = nt + \frac{n\,(10 + n)}{200} r$$

$$T = nt + \frac{n^2\, r}{200}, \text{ nearly.}$$

For holes of the following depths we have:

		ft.	ft.	ft.
n	=	400	800	1,200
t (minutes)	=	24	44	64
r (minutes)	=	7 1/3	8 2/3	10
T (minutes)	=	14,300	63,000	148,800
T (hours)	=	240	1,050	2,480

On Heinrich's work about 10 per cent. more time than the above was required to cover losses from delays arising from various causes. The point that is strikingly brought out by Heinrich's records is the rapid falling off in the rate of speed of drilling each foot of hole with increased depth. The cause is obvious, however, for the longer the line of drill rods the greater the friction of the rods upon the sides of the drill hole, and consequently the slower their revolution with an engine of limited horse power. The increased weight of the rods with increased depth also reduces the rate of speed with which they are hoisted by the engine; and this is a very important factor in adding to the labor and fuel cost of drilling deep holes. Heinrich's estimates of the time required to drill holes, including all 10 per cent. allowance for delays, are as follows:

400-ft hole,		288 hours
800 "	"	960 "
1,200 "	"	2,616 "

It will be observed that these times check fairly well with the times obtained by applying the formula that I have given; but it should be added that the constants in the formula need further verification by other observers. The material penetrated in the 800-ft. hole was:

Hard silicious sandstone..........	210 ft.
Medium " "	362 "
Argillaceous sandstone and slate.	237 "
Limestone	18 "
Total	827 "

Heinrich's estimates of time, and my own formula based thereon, assume a uniform sandstone throughout in the three holes. Had the rock been uniform throughout, the cost would have been:

400-ft.	hole, at	$1.26,	$504
800-ft.	" "	2.10,	1,680
1,200-ft.	" "	4.00,	4,800

Cost in Lehigh Valley.—Mr. L. A. Riley is authority for the following, as given in *Trans. Am. Inst. Min. Eng.*, 1876: Two machines belonging to the Lehigh Valley Coal Co. were used. A No. 2 drill with 16 H. P. boiler and 1,000 ft. of 2-in. rod cost $3,900, which with diamonds, etc., came to $5,000; the weight being 3,500 lbs. Carbons cost $9 per carat, and borts cost $11. Five diamonds weighing 18 carats were used per bit, drilling a 2-in. hole and bringing up a 1½-in. core. There were 24 holes, aggregating 9,902 ft., the deepest being 900 ft. The average rate of drilling these holes was 19 ft. per day per machine, at an average cost of $2.22 per ft. The rock was a very hard sandstone and conglomerate. The force on each drill was one foreman, one engineer and one fireman. The average cost per ft. of hole was:

Labor	$1.15
Diamonds	.66
Supplies and repairs	.41
Total	$2.22

The cost of the 900-ft. hole (the deepest) was $1.95 per ft., which indicates that with a powerful (16 H. P.) engine there is no such great increase in cost per ft. with increased depth as Heinrich found with an 8 H. P. engine. The 16-H. P. plant used by Riley was capable of drilling a 2,000-ft. hole. Note especially that both Riley and Heinrich paid less than $10 a carat for carbons and that Riley does not say what proportion of carbons to borts were used.

Cost on Croton Aqueduct.—Mr. J. P. Carson, in *Trans. Am. Inst. Min. Eng.*, 1890, gives the following:

Fourteen holes, total 2,084 ft., were drilled in the year 1895.

Actual days worked	189	days
Moving drill	15	"
Idle	18	"
Holidays and Sundays	39	"
Total	261	

		Daily Progress Feet.	Cost per ft.
347 ft.	Hard gneiss	11 to 12	$3.97
814 ft.	Decomposed gneiss	23.1 to 28	1.15
572 ft.	Clay, gravel and boulders	6.7 to 9	4.07
351 ft.	Clay and gravel	25	.....
2,084 ft.	Average	10.2	2.91

Crew, 1 foreman @ $125 mo.; 1 assistant foreman @ $70; 4 men @ $65.

Wages, 8.1 mos. $3,785

Team moving	$80
66.7 tons coal (189 days)	360
Supplies, Diamond Drill Co.	472
Foundry	291
Lumber, rope, etc.	53
Interest on $6,000 plant @ 12 per cent. 8.1 mos.	486
Renewing diamonds	250
Diamond bit lost	300
Total, 204 days	$6,077
Aver. per day	$29.79
Aver. per ft.	$ 2.91

Note that the interest on the plant is altogether too low.

Cost of Hand Diamond Drilling in Arizona.—*Eng. News*, Jan. 18, 1900, p. 34, Mr. J. B. Lippincott gives data on diamond drilling at the Gila River Dam site, Arizona. The machinery was in two distinct parts, (1) the hand pile driver for sinking casing pipe to bed rock; (2) the diamond drill. The hammer, made by the Pierce Well Co., 120 Liberty street, New York, is in sections, so that its weight can be varied up to 190 pounds; it is raised by a hand winch, and tripped by nippers; maximum drop 11½ ft. A tool-steel head is screwed into the top of the pipe and receives the blow. The pipe is 3½, 2½ and 2 in., extra heavy, screw pipe, 5 ft. sections, with extra heavy couplings which have beveled edges. When the casing has reached bed rock, the sand inside is removed by using a chopping bit and a water jet. The bit is screwed to a ¾-in. pipe through which water is pumped by a hand pump, the water passing out through holes in the bit, thus bringing the sand to the top of the casing. In this manner a casing pipe 130 ft. deep can be cleaned of sand and gravel. If a boulder is struck, after the diamond drill has penetrated it, four or five sticks of dynamite are lowered and discharged, shattering the boulder so that the casing can be driven down.

The diamond drill was made by the American Diamond Rock Drill Co., New York City. One inch core bits were usually employed. The drill was operated by hand power, six men being employed on this work as well as on driving the casing. The drill will penetrate 200 ft. into rock, and will make 6 to 8 ft. per day in hard rock and 10 to 15 ft. per day in soft rock. The plant complete costs $1,000, including two diamond bits worth $200 each, set with six 1-carat diamonds each. Two machines were used. The pipe cost $600 and freight, $100.

Cost of operation per month, foreman	$150.	
6 laborers at $1.50 for 28 days, ...	234.	
1 cook	45.	
		$429
240 rations at 60 cts.		144
Total labor for one month		$573
Total repairs, pipe and lumber for one party for 10 months		500
Team, feed, etc.		350
Moving		670
Sundry incidentals		430
Supervision		350
Total supplies, etc., for 10 mos.		$2,300
Total labor, 10 mos.		5,730
Total		$8,030
Total number of feet sunk		3,254
Cost per ft.		$2.46
52 holes, cost per hole		$154.42

Total Depths Penetrated.

	Earth, ft.	Rock, ft.	Total, ft.
The Buttes	1,621.2	196.0	1,817.2
Queen Creek	357.8	55.6	413.4
Riverside	729.8	40.2	770.0
Dykes	80.0	0.0	80.0
San Carlos	143.2	30.4	173.6
	2,932.0	322.2	3,254.2

A month's time of one party was lost due to continual breaking of the casing pipe under the hammer. Note that 90 per cent. of the drilling did not involve the use of diamonds but consisted in driving through the earth covering overlying the rock. This is characteristic, however, of testing dam sites.

References.—As stated in the fore part of this chapter, diamond drilling data are to be found in abundance. Among the most vauable of articles relating to the cost of diamond drilling are the following: In *Engineering News,* Apr. 2, 1903, data of drilling in South Africa; in *Engineering News,* July 23, 1903, is an excellent account of the cost of drilling test holes (100 ft. deep), along the line of a proposed canal in New York State; in the *Transactions* of the *Institution of Mining and Metallurgy,* Apr. 23, 1903, Mr. J. N. Justice gives valuable cost data of drilling deep holes (460 to 1,627 ft.), on the coast of Africa; in *Mines and Minerals* there are several excellent articles by Lane, in the last half of the year 1899; in *Engineering Magazine,* March, 1896, there is a good article by Channing, an engineer who has had much experience in Michigan; in the catalogues of American manufacturers of diamond drills much valuable information is given.

CHAPTER VII.

EXPLOSIVES.

The Action of Gases from an Explosion.—The explosives in common use in America are few in number: Black powder, several varieties of dynamite, Judson powder and Joveite. In every case an explosion is a chemical action that takes place between the elements of the explosive, liberating suddenly a great volume of gas at a high temperature. Ordinary air moving at a velocity of 100 miles an hour strikes objects with great disruptive force, yet a hurricane is mild indeed compared with the feeblest of explosives. Black powder, which is the weakest of the common explosives, is exploded whenever it reaches a temperature of 518° F. If its grains are small, as in rifle powder, each grain quickly burns up, yielding the full volume of gas in a short time. If its grains are larger the rapidity with which each grain is converted into gas is slower.

Dynamite and Jovite are exploded not by mere heating, as with black powder, but by a hard shock delivered usually by a cap, or "detonator," or "exploder." The more severe the blow delivered by the exploding of the cap, the more quickly does the chemical action take place. In any case this chemical action is far more rapid than it is in black powder, but the greater the shock the more rapid is the liberation of the gases. Hence it is poor economy to use feeble caps where it is desired to tear rock into small pieces.

If a quantity of dynamite is exploded under water it has been found by actual experiment that the gases fly in all directions with equal velocity and in equal quantity. * Hence it is a mistake to suppose that high power explosives act only downward. They exert a pressure equal in all directions at the instant of explosion. Let it be kept clearly in mind that an explosion is merely the sudden creation of a great volume of gas seeking to escape with enormous velocity, and much of the mystery about the effects of an explosion vanishes. Consider the gas as so many minute and invisible rubber balls, each possessed of weight and flying with frightful velocity, and it becomes comparatively easy to explain most of the phenomena attending an explosion. The gas as it leaves the explosive flies in all directions, but the instant it encounters an object it either rebounds from that object, or tears its way through the object, or hurls the object to one side. If the object is very heavy and substantial, the particles of gas rebound from it, like rubber balls rebounding from the side of a house. Hence it often happens that the gases from dynamite, when exploded in an open place, travel along a certain path like a hurricane. They do so because they have rebounded from some immovable objects, and have been, as it were, reflected like rays of light from a mirror.

---

* *Eng. News*, March 12, 1892, "D. E. O." in a letter states that in 1878, near Sawyer City, Pa., a wagon containing 60 qts. of nitroglycerine overturned, and the nitroglycerine exploded. The writer gives a sketch to show the wedge path of the exploded gases. The writer says: "Take two 2 or 3-oz. bottles and fill with nitroglycerine, fit one with an exploder and let the exploder rest on the bottom of the bottle; place this on a sheet of boiler iron and explode. It will be found to have barely brightened the plate. Now fit No. 2 with an exploder fastened in the neck of the bottle and explode, and it will be found to have blown a hole completely through the plate." The writer contends that this experiment proves that dynamite does not exert an equal pressure in all directions when exploded. The editor of *Engineering News* cites the extensive ring-gage tests conducted by Gen. Henry L. Abbot "with nitroglycerine under water." A 5-ft. ring carried six pressure gages, and to record pressure of dynamite fired at the center of the ring. The charge was placed in a tin can, placed vertically, and was fired at the top by a detonator. The ring was suspended in a vertical plane from buoys. At a depth of 35 ft., the two upper gages registered 20,366 and 17,576 lbs. per sq. in., and the two lower 15,479 and 14,671 lbs. In a later experiment, 16,000 and 21,000 lbs. were recorded in the upper gages; and 15,000 and 16,000 lbs. in the lower.

When dynamite is exploded in a hole drilled in solid rock, the gases are created so suddenly and move with such enormous velocity that when they strike the sides of the drill hole, the rock is struck as if with a mighty sledge; and, if the dynamite is in sufficient quantity, the blow tears off a portion of the rock. This occurs even when the hole above the dynamite is not plugged up with earth or stone chips; but if the hole is left open obviously much of the gas escapes and renders the blow less effective in consequence. Black powder, on the other hand, explodes more slowly so that if the hole is left open a greater proportion of the gas escapes before the last grain of powder has burned up. Hence it is absolutely essential to use great precaution in plugging the hole where black powder is to be fired.

In the early days of dynamite it was commonly stated that no "tamping" above the dynamite was required, and to this day there are text books in use containing this misleading statement. Tamping may not be "required," in the sense that it is absolutely essential; but it is certainly required where the full value of the dynamite is to be utilized in breaking rock, instead of disturbing the air, for there is no profit in shaking up the atmosphere.

Due to the fact that powder gases are elastic, and rebound from any solid surface, it follows that the shape of the drill hole has a decided effect upon the lines along which rock breaks. When holes are drilled by hand they tend to become three cornered; and as a result, if black powder is used, the particles of gas bouncing back and forth, as fast as they are liberated, batter the three faces of the triangle and tend to split the rock in three directions, the cracks in the rock starting at the corners, or angles of the triangle.

Black Powder.—The highest grade of black powder consists of 75 per cent. saltpeter (KNO_3), 15 per cent. charcoal and 10 per cent. sulphur. The charcoal for rifle powder (sporting powder) is commonly made from dogwood, but willow and alder charcoal are commonly used for blasting powder. In some inferior powders lampblack is substituted for part of the charcoal. The saltpeter (or nitre) is often replaced by sodium nitrate ($Na\ NO^3$), which deteriorates in time by absorbing moisture from the air. Therefore when blasting powder containing sodium nitrate ("soda powder") is used, great care must be taken to keep it in a dry atmosphere, and it should be used very soon after it is received from the factory. Powder is sold by the "keg" of 25 lbs., at about $1.25 a keg for soda powder and $2.10 for nitre powder. The specific gravity of individual grains of black powder ranges from 1.5 to 1.85; the average weight of loose powder, slightly shaken, being 62½ lbs. per cu. ft., or 1 lb. occupies 28 cu. ins.

Properties of Good Black Powder.—A good blasting powder has a uniform dark gray or slaty color. A dead black, or a bluish color, indicates either too much charcoal or the presence of moisture. When poured over a sheet of white paper it should leave no dust, for dust indicates either the presence of moisture or of fine mealy powder. The size of the grains should be quite uniform, and should have no sharp or angular corners. On pressing between the fingers there should not be a crackling sound due to sharp grains, nor should the grains crush easily. When crushed the grains should show the same uniformity of color. Light colored spots in the powder show that the saltpeter has leached out, due to the presence of moisture, which reduces the strength and reliability of the powder. If there are no white spots it may be assumed that the powder has not suffered from dampness; so that if it is slightly damp but still uniform in color it can be dried out in the sun and will be as good as ever. A pinch of good powder ignited on a sheet of white paper burns away rapidly, leaving no residue. If black spots remain on the paper they show an excess of charcoal or poor mixing of the ingredients. Yellow spots indicate an excess of sulphur. If holes are burned in the paper they indicate an excess of moisture, or other imperfections.

Dynamite and Nitroglycerin.—Any explosive containing nitroglycerin is commonly called dynamite. Nitroglycerin is made by mixing 1 to 1 1-6 parts of pure glycerin with 3 parts of nitric acid and 5 parts sulphuric acid. The glycerin is added very slowly, with constant stirring, compressed air usually being used to stir the liquids. The process of manufacture is exceedingly dangerous. Nitroglycerin is an oily fluid as clear as water when perfectly pure, but it usually has a yellowish tint. Its specific gravity is 1.6, so that it weighs nearly 0.058 lb. per cu. in., or 102 lbs. per cu. ft. It freezes at about 38° F. (water freezes at 32°), and instead of swelling as water does on freezing, it shrinks about 8 per cent. in volume. Nitroglycerin evaporates rapidly at 158° F.; and even at 104° dynamite will lose 10 per cent. of its nitroglycerin in the course of a few days. Hence the necessity of keeping dynamite in a cool place in summer, and in a warm, but not too warm, place in winter. In small quantities it will ignite and burn up without exploding at 356° F., but at 423° F. it explodes violently. In large quantities, heated slowly, it will explode at 356° F. If the nitroglycerin is impure it will explode at lower temperatures. Indeed it is possible for impurities to start a chemical decomposition which will result in a rise in temperature ending in spontaneous explosion.

If, after the mixture of the ingredients, every trace of acid is not washed out of the nitroglycerin, there is an ever-present danger of chemical action in the nitroglycerin that may lead to an explosion upon the slightest provocation. Chemical decomposition usually liberates nitrous fumes which color the nitroglycerin green. If there is any greenish color in dynamite it indicates that chemical action has begun and that the material is dangerous to handle. Free acid in nitroglycerin can be detected by blue litmus paper which the acid turns to red. In order to destroy deteriorated nitroglycerin pour it into a strong solution of sal soda (sodium carbonate) and stir gently with a wooden paddle.

Pure nitroglycerin has been carried by a rocket to a height of 1,000 ft. and dropped without exploding upon striking the earth. Yet the purest of nitroglycerin is liable to explode by shock if it is confined in a vessel. When impure it will explode, even when unconfined, upon receiving a slight shock. A small quantity of nitroglycerin will burn quietly without exploding; but where a large quantity is burning the heat generated will bring the entire mass to a temperature at which an explosion will occur.

On account of its sensitiveness to shock when slightly impure, nitroglycerin is not used for blasting to any great extent nowadays. It is used in its liquid state chiefly for "shooting" oil wells, so as to open up crevices in the rock through which the oil may flow to the well. The nitroglycerin is poured into tin "shells," 3 to 5 ins. diam. by 5 to 20 ft. long, and lowered with a wire to the bottom of the well hole. An iron weight with a hole through its center is strung on the wire and allowed to drop, thus exploding a cap on the cover of the "shell."

Varieties of Dynamite.—Dynamite consists of any absorbent or porous material saturated or partly saturated with nitroglycerin. The absorbent is commonly called "dope." A good dope should have minute voids in which the nitroglycerin is held by capillary action. Since the dope acts like a cushion it renders the nitroglycerin much less sensitive to shocks. If 40 per cent. of the weight of the dynamite is nitroglycerin it is known as a "40 per cent. powder"; if 75 per cent. it is a "75 per cent. powder." The word "powder" is commonly used instead of the word dynamite, and, in consequence, it often confuses the hearer or reader who is at a loss to know whether black powder or dynamite is meant.

The following are some of the well-known dynamites:

ATLAS POWDER (75 per cent.)

Nitroglycerin	75	parts.
Wood fiber	21	"
Sodium nitrate	2	"
Magnesium carbonate	2	"

RENDROCK (40 per cent.)

Nitroglycerin	40	parts.
Potassium nitrate	40	"
Wood pulp	13	"
Pitch	7	"

GIANT POWDER NO. 2 (40 per cent.)

Nitroglycerin	40	parts.
Sodium nitrate	40	"
Sulphur	6	"
Resin	6	"
Kieselguhr	8	"

STONITE (68 per cent.)

Nitroglycerin	68	parts.
Kieselguhr	20	"
Wood meal	4	"
Potassium nitrate	8	"

DUALIN (40 per cent.)

Nitroglycerin	40	parts.
Sawdust	30	"
Potassium nitrate	30	"

CARBONITE (25 per cent.)

Nitroglycerin	25	parts.
Woodmeal	40½	"
Sodium nitrate	34	"
Sodium carbonate	½	"

HERCULES (40 per cent.)

Nitroglycerin	40	parts.
Potassium nitrate	31	"
Potassium chlorate	3 1-3	"
Magnesium carbonate	10	"
Sugar	15 2-3	"

VIGORITE (30 per cent.)

Nitroglycerin	30	parts.
Potassium chlorate	49	"
Potassium nitrate	7	"
Wood pulp	9	"
Magnesium carbonate	5	"

HORSLEY POWDER (72 per cent.)

Nitroglycerin	72	parts.
Potassium chlorate	6	"
Nutgalls	1	"
Charcoal	21	"

GELIGNITE (62½ per cent.)

65 per cent. of blasting gelatin, containing	Nitroglycerin, 96 per cent. Collodion cotton, 4 per cent.
35 per cent. of absorbent, containing	Sodium nitrate, 75 per cent. Sodium carbonate 1 per cent. Wood pulp, 24 per cent.

FORCITE (49 per cent.)

50 per cent. of blasting gelatin, containing	Nitroglycerin, 98 per cent. Collodion cotton, 2 per cent.
50 per cent. of absorbent, containing	Sodium nitrate 76 per cent. Sulphur, 3 per cent. Wood tar, 20 per cent. Wood pulp 1 per cent.

JUDSON GIANT POWDER No. 2 (40 per cent.)

Nitroglycerin	40	parts.
Sodium nitrate	40	"
Resin	6	"
Sulphur	6	"
Kieselguhr	8	"

VULCANITE (30 per cent.)

Nitroglycerin	30	parts.
Sodium nitrate	52½	"
Sulphur	7	"
Charcoal	10½	"

Many of the "powders" above named are made with different percentages of nitroglycerin. A "No. 1 powder" ordinarily contains 75 per cent. of nitroglycerin; and a "No. 2 powder," 40 per cent. nitroglycerin; but the manufacturers have a great variety of letters and numbers to denote the different grades of "powder." The following table indicates how varied is the numbering and lettering of the different grades made by different firms:

Aetna Powder, No. 1	65	per cent.
" " No. 2XX	50	"
" " No. 2	40	"
" " No. 3X	35	"
" " No. 4X	25	"
" " No. 5	15	"
Atlas Powder A	75	"
B+	60	"
B	50	"
C+	45	"
C	40	"
D+	35	"
D	30	"
E+	25	"
E	20	"
F+	15	"
Dynamite (Nobel's) Old No. 1	75	per cent.
" " " No. 2	40	"
" " " No. 3	25	"

The Absorbent.—Alfred Nobel, who invented dynamite in 1866, used porous, earthy powder, called kieselguhr, as the absorbent to hold the liquid nitroglycerin in its pores, somewhat as a sponge holds water. Kieselguhr is a diatomaceous earth which consists of the silicious remains of microscopic plants called diatoms. These diatoms contain microscopic pores or cells which hold the nitroglycerin by capillary action. Of late years sodium nitrate and wood pulp have been very largely substituted for kieselguhr.

I find in the Census Report for 1900 that in the annual production of 42,900 tons of dynamite there were used: 15,800 tons of nitroglycerin, 20,000 tons of sodium nitrate, 5,000 tons of wood pulp and 240 tons of ammonium nitrate. It will be seen from this report that dynamites with an inert base or "dope" of kieselguhr, or magnesium carbonate, are no longer made in America; and that dynamites with an explosive base of the nitrate class have taken their place. It will also be seen that the nitroglycerin used averages about 38 per cent. of the weight of the dynamite as now manufactured.

Weight of Dynamite.—Dynamite is commonly packed in paper cartridges, each "stick" being about 6 to 10 ins. long and of a diameter to slip readily into the hole. The most common size of stick is 1¼ ins. diam. by 8 ins. long, weighing ½ to 6-10 lb. Dynamite is shipped in cases, or boxes, holding 50 lbs. of dynamite, and must be packed and marked in accordance with the rules prescribed by the railroad over which it is to travel. Not every user of dynamite knows that an act of Congress (1851) provides that:

"Any person or persons shipping explosives without delivering at the time of shipment a note in writing expressing the nature and character of such merchandise shall forfeit to the United States $1,000."

The old Nobel's 75 per cent. dynamite, with the kieselguhr "dope," weighs .054 lbs. per cu. in. The following table gives the weight of 75 per cent. keiselguhr dynamite:

Weight of Nobel's No. 1 dynamite (75 per cent. nitrogen and 25 per cent. kieselguhr):

Diam. of stick.	Weight in lbs. per inch of stick.	Diam. of stick.	Weight in lbs. per inch of stick.
1 in.	.042	1¾	.128
1¼ ins.	.065	2	.168
1½ ins.	094	2¼	.212

The manufacturers of "Atlas C" inform me that a 1¼ x 8-in. stick weighs about 0.6 lb., and that it is one of the "tricks of the trade" to make the absorbent of such material at all grades of dynamite weigh about the same per stick (1¼ x 8 in.).

Thawing Dynamite.—The nitroglycerin in dynamite freezes at 42° to 46° F., according to the character of the "dope." When frozen it cannot be exploded by the ordinary caps used in blasting; nevertheless in its frozen state it is exceedingly sensitive to friction or to any breaking or cutting of the frozen cartridge. The Annual Report for 1898 of the Inspectors of Explosives of Great Britian states that in 1898 there were 81 accidents in thawing dynamite, resulting in killing 68 men and injuring 97. Accidents from other causes were 194 in number, resulting in the killing of 52 men and the injury of 216. This shows in a striking manner how dangerous a process the thawing of dynamite is. The following are a few of the methods of thawing that are given in the report as having led to injury or death:

Some cartridges being thawed on a stone in a weigh house; thawing cartridges in front of a kitchen fire; thawing dynamite on a shovel; cartridges placed near a fire to thaw; cartridges placed in an oven to thaw; hot-water thawer, containing dynamite placed on a blacksmith's fire; thawing dynamite with a candle; warming dynamite over a blacksmith's fire; heating dynamite in a tin over a candle; rubbing cartridge in hands to complete thawing; cartridge left in pocket of trousers, which were hung before fire to dry; thawing dynamite in water over a fire; *nine separate accidents from reheating water which had been used in a dynamite thawer,* averaging one killed and one injured for each accident.

I have italicized this last mentioned cause of accident, because in my judgment it is at the base of the great majority of all accidents from thawing dynamite. It shows that the nitroglycerin leaks out of the dynamite especially when subject to heat in the presence of water. Dynamite should never be thawed by plunging the sticks into warm water. Nitroglycerin will often leak out of a stick even when the stick is dry but is subjected to heat. This I have actually seen. In driving a prospect tunnel in Idaho it was our custom to thaw dynamite in the oven of a cook-stove; but, when the thawing did not progress rapidly enough, we would hold sticks of dynamite in our hands over the top of the stove. While doing this one day I noticed that a drop of nitroglycerin had leaked through the folds of the paper cartridge and was about to fall upon the stove. In my haste to move the stick away the drop fell upon the hot stove, exploding with a report like a pistol shot and cracking the stove so badly that live coals fell into the oven. The dynamite in the oven immediately began to burn up without exploding. It is perhaps needless to add that those of us who were in the cabin went out during the time that the dynamite was baking. As stated in previous paragraphs, free nitroglycerin is exceedingly sensitive to shock and heat combined, and one drop of it falling even a short distance upon any hot object will explode, and by its explosion set off any sticks of dynamite nearby.

Mr. E. E. R. Tratman has cited an instance where dynamite sticks were placed on a canvas cover over a pot of boiling water; nitroglycerin leaked out, and through the canvas settled on the bottom of the pot, where it exploded, the water above it acting as a tamping.

A man working for me laid some sticks of 75 per cent. dynamite upon a flat stone which he had previously heated by placing hot coals upon it. While in the act of picking up a handful of thawed sticks he was blown to atoms. Without doubt the cause was the leaking out of a drop of nitroglycerin which, falling upon the hot stone, exploded the remaining sticks. He was using this method directly contrary to orders, because he had "thawed dynamite all his life in that way." Familiarity breeds contempt for the danger ever present in thawing dynamite, and the manager of blasting operations must not rely merely upon orders to the men not to do this or that, but must be vigilant to observe whether orders are obeyed or ignored. Instant discharge of an employee should be the punishment for the slightest infraction of rules governing the use of explosives.

Dynamite can be ignited with a match, and will usually burn up without exploding, provided that there are only a few sticks not confined in any way. This fact has had much to do with breeding contempt for the danger attending thawing. Low-grade dynamites (40 per cent. and under) are safer to thaw than high-grade dynamites, because the "dope" is not so thoroughly saturated with nitroglycerin, and for that reason is not so apt to "leak"; but in the presence of hot water any grade of dynamite will have its nitroglycerin displaced slowly by the water. In fact, if the manufacturers are not careful to remove every trace of water from the nitroglycerin there is danger of "leaking." When the paper cartridges feel greasy it is due to leakage of nitroglycerin. When a whitish crust, or efflorescence, is found on the outside of a dynamite cartridge it indicates that the dynamite has been stored in a damp place, or that the "dope" originally contained an excess of moisture. In either case the crust is nitrate of soda that has dissolved out, and such dynamite is almost certain to leak nitroglycerin. It is unreliable, dangerous to handle, and should be destroyed at once. Greenish stains inside the cartridge indicate that the nitroglycerin is decomposing and is dangerous.

I have laid particular stress upon the leaking of nitroglycerin from dynamite, because it is so common a source of accident and because the fact that dynamite can be exploded under water has lead many to infer that water has no deleterious effect upon it. Dynamite under water begins to part with its nitroglycerin immediately, the water slowly replacing the nitroglycerin in the "dope," and even in cold water a few hours of soaking will materially decrease the percentage of nitroglycerin. In warm water the replacement is much more rapid.

How to Thaw Dynamite.—We have seen how dangerous it is to thaw dynamite by plunging the sticks into hot water or by allowing live steam to strike the sticks, due to the fact that the water forces the nitroglycerin out of the stick; and we have seen how sensitive such free nitroglycerin is to slight shocks, especially when hot. We have seen how exceedingly dangerous it is to place dynamite upon a stove, or in front of an open fire, or upon the top of a steam boiler, or upon a hot stove. How, then, can dynamite be thawed with comparative safety?

Green manure is an effective and safe material to use in thawing dynamite. On the Croton Dam work the following method was used: A cubical box 2½ ft. on a side is set inside a box 16 ins. larger on a side, and the 8-in. space filled with manure rammed hard. These two boxes are placed in a cubical hole in the ground and 15 ins. of loosely rammed manure is packed around. The floor of this magazine is filled to a depth of 10 ins. with hard rammed manure, leaving a remaining space that easily holds 50 lbs. of dynamite. The lid is provided with a pipe chimney 2½ ft. long, having a sliding cover for ventilation. The dynamite is piled in loosely, the lid closed and manure covered over it to a depth of 12 ins. The ventilator, as a rule, is left slightly open, and at 32° F. outside the powder will thaw in 3 to 5 hrs.; at 0° F. outside it will thaw in 8 hrs. This thaw box is cheap, simple and safe. The manure on the bottom of this magazine acts as a cushion to absorb any nitroglycerin that might leak out. It is necessary to change all the manure occasionally.

I am not favorably impressed with any method of thawing that involves standing the sticks of dynamite on end, for that facilitates the leakage of nitroglycerin. I have seen a box of dynamite that had been stored on its end in a magazine, and the nitroglycerin had leaked from the cartridges saturating the wood of the box.

The plan of placing a can of hot water in a small thawing magazine is one of the safest methods that can be adopted. Such a method is illustrated in Fig. 7, page 120.

Where a very large quantity of dynamite must be thawed daily, a small thaw-house should be built with several doors

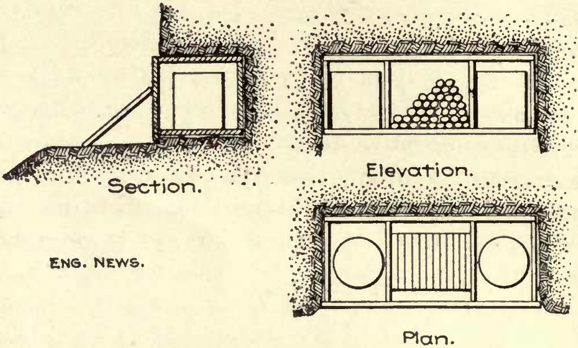


Fig. 7.

in front, and tiers of drawers that slide out should be placed immediately back of the doors, so that a man cannot enter the thaw-house itself from the front. This prevents men from loitering in the thaw-house, and possibly standing inside to light a pipe. The dynamite is laid in the drawers (6 ins. deep x 16 x 22 ins.) on a thin bed of sawdust. In the rear of the thaw-house, back of the drawers, room is left for a small hot water radiator such as is used in house heating; 1-in. pipes lead from this radiator to the hot water heater which is some distance away from the thaw-house, in a separate building entirely, so that there is no chance for the thaw-house to be set on fire. Under no condition use steam to heat the thaw-house. A temperature greater than that of boiling water (212°) should not by any possibility be reached inside the house.

The foregoing are the only methods of thawing dynamite permitted by the Municipal Explosives Commission in New York City, namely, thawing with manure, and thawing in a dry chamber heated by hot water entirely separate from the fire that heats the water.

Testing Dynamite for Safety.—The outside of dynamite cartridges should not feel greasy, nor should there be a trace of free nitroglycerin inside the wrapper. In order to determine whether a stick is leaky, dry it on a clean sheet of brown paper in a room at 60° to 80° F. for about 12 hours. An oily discoloration on the brown paper shows that nitroglycerin has leaked out. Good dynamite will show no such discoloration of the paper.

Dynamite that has been frozen and thawed a number of times often leaks, although before the freezing and thawing it did not leak at all. Hence a few sticks should be frozen and thawed three successive times and then tested for leakiness on brown paper as above explained.

Long-continued, high temperature will develop leakiness in a poor quality of dynamite. Hence a few samples should be kept at a temperature of 85° to 90° F. for six consecutive days and nights, and then tested for leakiness on brown paper as above explained.

A whitish crust on dynamite sticks indicates that it has been damp and that the nitrate of soda or potash has leached out (effloresced); and that consequently the dynamite is no longer reliable, and may fail to explode in blasting, beside being dangerous to handle. If the dynamite inside the wrapper shows greenish spots it indicates decomposition of the nitroglycerin, and consequently is exceedingly dangerous.

Blasting Gelatin.—Soluble guncotton dissolved in nitroglycerin gives a jelly-like substance of yellow color known as blasting gelatin. It has a specific gravity of 1.6 and freezes at 35° to 40° F. as compared with 42° to 46° at which dynamite freezes. It is far more dangerous than dynamite when frozen, being more sensitive to shocks in the frozen condition than when soft. It is peculiarly adapted for use in tropical climates or in summer work, since it does not absorb water and does not leak under any conditions, even after long exposure to 90° F., nor does it leak after repeated freezing and thawing. In cold weather its extreme sensitiveness when frozen makes it exceedingly dangerous.

Gelatin dynamite is an explosive containing blasting gelatin and an explosive "dope." Forcite and gelignite are the two best known gelatin dynamites. They are apt to leak and should be tested precisely as ordinary dynamite is tested for leakage by repeated freezing and thawing and by prolonged exposure at 90° F.

Judson Powder and Contractors' Powder.—Judson powder consists of:

Nitroglycerin	5 per cent.
Sodium nitrate	64 " "
Sulphur	16 " "
Cannel coal	15 " "

The "contractor's powder" made by the Aetna Powder Co. is similar, but it contains 8 per cent. nitroglycerin. They are both free running black powders made honey-combed so as to hold a small percentage of nitroglycerin, and are fired with caps or exploders exactly as dynamite is fired. They are sold in 50-lb. boxes.

Joveite.—Joveite is a free-running, dry powder resembling corn meal in appearance. It does not freeze, nor has it ever been found to deteriorate upon exposure to heat. It is composed of nitro-napthalene, nitro-phenol and nitrate of soda. Recently the manufacturers have succeeded in making it perfectly waterproof. I have seen it charged in holes full of water and exploded without a misfire. The manufacturers have made strong claims not only for the effectiveness of this powder in competition with dynamite, but for the far greater safety attending its use. The fact that it does not freeze alone entitles it to consideration as a "safety explosive," but it can also be fired into with rifle bullets, hammered with iron on iron, burned up by lighting it with a match, heated to the burning point with electric sparks from a powerful static electric machine, and otherwise subjected to the most severe tests without exploding. It will burn up quietly when a certain temperature is reached, but it takes a strong cap, placed not further than half an inch from the stick of explosive, to explode it. I have personally subjected Joveite to these and other tests for safety, and am convinced that it is incomparably the safest, reliable explosive in the market to-day. Joveite has been in extensive use for about five years, and as yet no accident has occurred in using it. I have fired an iron projectile from a mortar loaded with a weighed quantity of Joveite, and have found it superior to dynamite in hurling power, grade for grade. Joveite is manufactured in four grades, each grade being equal to the various grades of 20 per cent. to 60 per cent. dynamite.

I have inhaled the gases resulting from exploding Joveite without suffering from the headache that follows such a test with dynamite; from which it would appear that it would be less objectionable than dynamite in underground work.

Joveite can be poured into dry holes like Judson or black powder, but ordinarily it is charged in paper cartridges exactly as dynamite is charged, and fired by the same kind of caps or exploders. In its action, Joveite is a trifle "slower" than dynamite, but when the higher grades are used it breaks the rock into smaller fragments.

On the great rock excavation work now in progress at the New York Central yards in New York City I find that Joveite has displaced dynamite and is doing more efficient work in the tough rock encountered there, and at less cost than dynamite. The same holds true of several of the large trap rock quarries near New York City. I am able, therefore, to agree with Prof. Courtney De Kalb, of the School of Mining, Kingston, Ontario, who, in his excellent little "Manual of Explosives," says: "Joveite has been tested by the ablest explosive experts and has never proven unsafe or unreliable. It would seem to fulfil all the requirements of an ideal explosive."

CHAPTER VIII.

CHARGING AND FIRING.

Kind of Explosive to Use.—Whether a high power or a low power explosive is to be preferred is dependent largely upon the use to which the rock is to be put, as well as upon the strength of the rock itself. Black powder, with its comparatively slow, heaving action, is used where the material is quite friable, as in mining coal or galena, or in excavating shale, hardpan and the like. It is also used in small charges placed in a row of holes where it is desired to wedge off blocks of "dimension stone" for building purposes.

Judson powder (which contains a small percentage of nitroglycerin) is considerably more powerful than black powder, and is used in open cut excavation where the rock is of medium strength. It is also used in "chamber blasting," where large charges of it are placed at the end of a small tunnel and a mountain of rock dislodged at one shot. In such cases it will break up very hard rock, leaving it, however, in large chunks.

A high power explosive like dynamite is invariably used in tunnel driving, shaft sinking and open-cut work in tough rock. Specifications usually prohibit the use of dynamite for quarrying dimension stone, because it is apt to shatter the stone. For quarrying stone to be used as rubble, especially if the stone is tough and occurs in massive layers, dynamite can usually be used without danger of injuring the stone. A 40 per cent. dynamite is commonly used in open cut work, but with tough rock it often pays to use a 50 to 75 per cent. dynamite (or the equivalent grade of Joveite) especially if the rock is to be shattered so that it will pass through a crusher or is to be loaded with a steam shovel. I have found it advantageous to begin blasting in open cuts by using 40 per cent. dynamite. If the rock comes out in too large chunks then to every three sticks of 40 per cent. powder I use one stick of 75 per cent.; and in successive blasts increase the proportion of 75 per cent. until the rock comes out in chunks of desirable size. Experiments should also be made in spacing the holes, but of this I will speak more at length later. Having found the proportion of 40 per cent. to 75 per cent. dynamite yielding the best results, it is possible to order a grade of dynamite that will contain the desired percentage of nitroglycerin. Thus, assuming that the best charge is two sticks of 40 per cent. to one stick of 75 per cent., we have:

$$\begin{array}{rr} 2 \times 40 \text{ per cent.} = & 80 \\ 1 \times 75 \text{ per cent.} = & 75 \\ \hline 3 \qquad\qquad\qquad & 155 \end{array}$$

$155 \div 3 = 52$ per cent., which is approximately the grade of powder to order. If the job is small, one can continue to use a mixture of 40 per cent. and 75 per cent. dynamite, but on large work it involves too much trouble to use two grades of powder in the same hole. Moreover, the 75 per cent. dynamite is far more dangerous to handle, particularly where it must be thawed. Managers and foremen are prone to do all their experimenting by changing the spacing of the drill holes or by changing the weight of the explosive used in the charges, instead of experimenting to determine the most effective grade of explosive to use.

In tunneling, the "cut holes" are frequently charged with 75 per cent. dynamite, and the "trimming holes" with 40 per cent. dynamite. In tunneling through weak rock, like shale, 40 per cent. dynamite will be found powerful enough even for the "cut holes." Vast sums of money are daily wasted in the mines and quarries of the United States through lack of systematic experimenting to determine the most economic grade of explosive and the most economic spacing of drill holes. By taking the work of blasting temporarily out of the hands of my foreman I have repeatedly succeeded in reducing the "powder bill" from 10 per cent. to 35 per cent., and I do not hesitate to say that a foreman who can be trusted to select the proper grade of explosive intellegently is "one in a hundred."

Charging Black Powder.—After pumping out the sludge, the hole is made perfectly dry by a "wiper," using cotton waste or hay held by a spiral twist at the end of the "wiper." The other end of the "wiper" is often provided with a small spoon for scraping out the sludge at the bottom of the hole. If the hole is a small one the powder may be poured through a tin funnel with a long stem reaching to the bottom of the hole, so that none of the powder lodges upon the sides. In large, deep holes no such precautions are taken. If the hole is horizontal the powder may either be shoved in in paper bags, or a long spoon-like scoop may be used to deliver the powder to the end of the hole where it is dumped by revolving the handle of the scoop. A safety fuse should be used (or electric cap), and its lower end should be well buried in the powder. If paper cartridges are used, the end of the fuse is shoved into the powder and the paper tied around it; but do not pull the string so tightly as to pinch the fuse so as to break the powder thread inside it. If the hole is a wet one, a waterproof cartridge must be used. To make such a cartridge, fold a long strip of brown paper spirally around a wooden mandril, slightly smaller than the diameter of the drill hole, at its lower end, letting the edges of the paper overlap well. Before removing the paper from the mandril, dip it into melted paraffine, giving it several coats. In a very wet hole another spiral paper wrapping in the reverse direction, well paraffined, will insure dryness. Load this cartridge with powder, attach the fuse and immerse in melted paraffine (113° F.). This cartridge will be perfectly water tight, but cannot be rammed.

After the drill hole is loaded a tamping of clay or sand is used to fill the hole. The kind of tamping has a very great effect in determining the force of the explosion of black powder. Clay is unquestionably better than sand and should be used for the first few inches anyway. Dry clay is first pressed down with a wooden tamp rod. Never use a metal rod and never ram the tamping at the start, for fear of an explosion. Follow with ordinary damp clay pressed firmly to place, and after a thickness of three inches of tamping is over the powder, ram by tapping the end of the tamping rod with a hammer. In holes 1 in. in diam. the charge will not blow out if there are 7 ins. of good tamping. In general the tamping will not blow out if it is 7 to 10 times as long as the hole is wide. Nevertheless the tamping should be carried to the surface of the rock if the greatest effect of the powder is desired. If there are any spaces between the powder and the sides of the hole, or between the powder and the tamping, the effect is to cushion the blow of the explosion. In quarrying dimension stone this cushioning effect is sometimes desirable, and it is purposely secured by filling several inches of the hole above the powder with hay, tow or the like, followed by several inches of clay tamped lightly, and finally by well packed tamping. This is called "expansion tamping."

In firing black powder by electricity, electric exploders of low power are used. There is no advantage in using powerful detonators, because black powder cannot be detonated, but explodes in the same way whether a match or an electric spark or another explosive fires it.

Charging Dynamite.—The charge should fill completely the part of the hole that it occupies, and should be packed solid. Experiments show that even a slight air cushion greatly weakens the force of the explosive blow. Since the sticks of dynamite are slightly smaller in diameter than the hole, the paper of the cartridges (excepting the last one) should be slit lengthwise with a knife * and after each stick is dropped or pushed into the hole, press it well home with a wooden rammer. If there is standing water in the hole do not break the paper of the cartridge, and do not ram, but use a cartridge that will just fill the hole. In wet holes it is well also to daub grease over the cartridge wherever water might enter through a fold in the paper. The cartridges should never be so large as to require forcing to get them to the bottom of the hole. Remember that a drill hole tapers toward the bottom. Dynamite should never be rammed, but merely pressed home; and a steel or iron tamping rod should never be used for this purpose. The last stick, or "primer," is provided with either a fuse cap or an electric detonator. If a fuse is used a common way of loading is first to slip the end of the fuse into the cap, bite the end of the cap shell so as to pinch it upon its fuse; and, if the blaster survives this part of the operation, the next step is to dig a hole in the middle of the dynamite stick with a wire nail; push the cap into the hole and pinch the plastic dynamite around it; take a half-hitch with the fuse around the dynamite cartridge and lower it or push it to place. A cap should never be crimped onto the fuse with anything but a "crimper" made for the purpose. A half-hitch in the fuse is quite apt to break the powder thread inside the fuse and thus cause a misfire. If an electric exploder is used, taking a half-hitch with the fuse wires is apt to result in breaking one of the wires away from the platinum bridge to which it is soldered, and thus causing a misfire. An expert, however, who is skilful and careful may use the half-hitch without causing misfires. The method recommended in all catalogues of manufacturers is first to open the end of the

* Never cut with a knife or otherwise rupture a stick of dynamite that is frozen or partly frozen. Some authorities recommend using a copper blade instead of steel, because the steel might strike a spark if there were any grit in the cartridge.

"primer" cartridge by folding back the paper; then to insert the cap part way into the dynamite after boring a little hole in the dynamite with a wooden stick with a rounded point. The cap is left projecting about ⅛ in. above the dynamite, so that by no chance can the fuse set fire to the dynamite and thus reduce the force of the explosion. The ends of the paper cartridge are drawn up around the fuse or the fuse wires, and tied with a string as in Fig. 8, one end of the string being left long enough to let the "primer" down to the bottom of the drill hole. In any case the cap should fit tightly in the dynamite, for even a slight air space will serve as a cushion to reduce its force and so weaken the force of the final explosion. In wet holes, smear grease around the end of the cap. The end of the fuse should be cut square across, preferably with a "fuse cutter," and then, holding the fuse upright, slip the cap over it. It should require no effort at all to slip the cap on, for either pressing or twisting the cap on may explode it. If the fuse is too large whittle it down with a knife; if too small, wrap paper around it. Crimp the shell of the cap about ⅛ in. from its end with the "crimper," which is combined with the "fuse cutter." At the other end of the fuse cut a slit ½ in. long to expose the powder core for lighting with a candle flame or torch. Dry paper may be twisted around the end to insure lighting, but it is not good practice to soak cloth or waste in oil and wrap it around the fuse. Of course the end of the fuse should not be allowed to drop into water, at least not until the fire inside has crept some distance down into the fuse. The "primer" should not be lowered into the hole by the fuse, because in this way the cap is often pulled loose leaving an air cushion that greatly reduces the force

Fig. 8.

of the explosion. When the "primer" has been lowered it should never be compressed or rammed; but the tamping should be placed upon it immediately.

To emphasize the importance of inserting a cap and fuse in the end rather than in the side of a stick, a quotation from a paper by Mr. A. W. Warwick in *Mines and Minerals* will serve:

"There was no doubt in my mind, after studying the method of loading, that there was a possibility of the burning fuse setting fire to the dynamite cartridge on top of the primer before exploding the cap. In order to see if this were the case or not, a piece of pipe 12 in. in length and ¾ in. in diameter was obtained, a piece of fuse was passed through and a cork was forced in so as to hold the dynamite; a stick of dynamite was squeezed into the pipe and held in place by a plug. The fuse was fired and developments were awaited at a respectful distance. Out of seventeen experiments six resulted in an explosion. The fumes from the explosion were very acrid, dense and rather ruddy in color. The nitroglycerin was fired by heat and not by detonation, and the fumes had the appearance and odor of fumes generated by incomplete combustion."

The best tamping is dry clay or bits of shale, and even where sand is used for the major portion of the hole it will pay to use clay balls for the first foot or so, the clay being moist enough to roll into pellets. A handful or two of sand may be poured into the hole first to cover the "primer"; and then follow with clay. The clay pellets should be lightly compressed for the first 6 ins., and above that the tamping may be compressed with increasing force. Sand is generally used for tamping above the first foot or two because it can be poured in with much greater rapidity.* Experimenting with different kinds of tamping on any given class of work is time and money well spent, for it is not a fact that dynamite needs no tamping, or that water makes a good

* I would suggest pouring enough water into the hole after the sand is in to dampen it, for damp sand arches better than dry sand and better resists the pressure.

tamping. Mr. W. L. Saunders is authority for the statement that one pound of dynamite under a water tamping will not do as much execution as one-quarter of a pound in dry blasting. Bear in mind that tamping is cheaper than dynamite even if several dollars a ton are paid for tamping imported by rail. While working the tamping rod with the right hand, hold the fuse or the fuse wires with the left hand so as to detect and thus avoid rubbing the fuse with the tamping rod. Some blasters use a tamping rod with a beveled end, and hold the rod so that the sharp edge of the bevel is always on the side of the hole farthest from the fuse. To know which side the sharp end is on, cut a longitudinal groove in the tamping rod.

Some authorities recommend placing the "primer" at the bottom of the charge, instead of at the top; others say that the "primer" should be placed at the middle of the hole. Dynamite explodes with such suddeness that we may well doubt whether it makes any difference at all where the primer is placed, so far as the execution is concerned. It is often advisable, in deep holes, to place the dynamite in several distinct charges separated by tamping, and in this case each charge should have its own cap and "primer"; but this is a matter quite aside from the present discussion and will be taken up later.

Handling dynamite sticks with the bare hands will give a headache to anyone not used to it, because of the nitroglycerin absorbed through the pores of the skin. The obvious preventative is to wear gloves.

Judson powder is charged like black powder, but it is fired by using a "primer" consisting of a stick of dynamite in which a blasting cap is imbedded. Joveite is charged and fired like dynamite.

Firing by Electricity.—In New York City it is now compulsory to fire all blasts by electricity on account of the greater safety of electric firing. Electric firing is not only safer than fuse firing, but, in open cut work especially, it is more effective, because the simultaneous explosion of charges in a row of holes obviously reduces the work to be done by each charge as compared with fuse firing by which one charge explodes in advance of the neighboring charge. In tunneling, where the center cut holes must be fired in advance of the outer holes, it will probably continue to be the practice to fire by fuse, using fuses of different lengths so as to regulate the order in which the charges in the different holes will explode, but there are few places outside of tunnels and shafts where fuse firing is preferable to electric firing from any point of view; and even in tunnel work there are many blasters who prefer electric firing.

For firing by electricity the electric detonator, A (Fig. 9), is placed in the "primer," and the fuse wires, B, are carried up along the "primer" and tied with a cord, C, to the "primer." The fuse wires attached to each cap are furnished by the manufacturers in varying lengths to suit varying depths of drill holes; but it is not necessary to have fuse wires that will reach to the mouth of the drill hole, for connecting wires may be spliced on to the ends of the fuse wires and the splice wrapped with insulating tape. When a splice is to be made thus, it is well to cut 3 or 4 inches off one of the fuse wires so that one splice will not come directly opposite the other; for when this is done it is necessary to wrap insulating tape around one of the splices only, and it is not necessary to wrap even one splice except in damp holes. A splice is made by cutting the insulating material away for 2 or 3 ins. back of the ends of the wires to be joined, scraping the wires until they are bright, and twisting the clean wires together, as shown in Fig. 10. Insulating tape

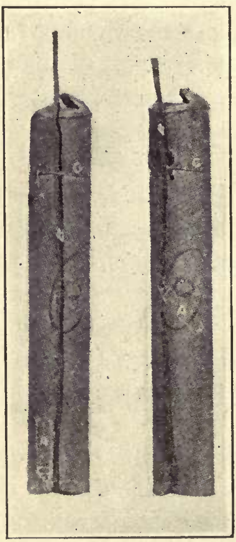


Fig. 9.

is ½ to ¾ in. wide and comes in ½-lb. rolls. It is often used to cover splices made in connecting wires when the splice comes in contact only with dry rock on dry

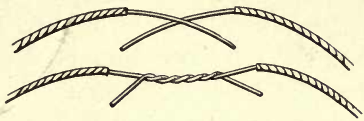


Fig. 10.

earth. Practically no electricity can leak through dry rock or earth, so that there is no necessity of insulating the splices unless the bare wire is apt to come in contact with moist earth or water. The fuse wires from each hole are connected right and left with the fuse wires from the neighboring holes, so as to form a continuous circuit with the holes in series (Fig. 11); then the end of the fuse wire of the hole on the extreme left is connected with a "leading wire" that runs to the electric

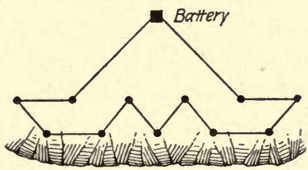


Fig. 11.

battery, and in like manner the other "leading wire" from the battery is connected with the fuse wire of the hole on the extreme right. This final connection should not be made at the battery until all workmen are at a safe distance. When it is made, lift the handle of the battery and press it down, at first with moderate speed, but finishing with full force. The leading wires must be long enough so that the electric battery is 200 to 500 ft. away from the blast, and in a direction back of the face (in open cut work). The sun should not shine in the eyes of the blaster, for he should be able to see and dodge any falling fragments of rock. It is well to wind the two leading wires together into one cable, separating them only for a short distance from the blast; then, after firing, this cable can be wound up on a reel. As the copper connecting wires are expensive, the blaster should collect all fragments of wire after each blast and wind them upon a reel to be spliced and used again.

Misfires.—A misfire when an electric battery is used may be due to any one of several causes: (1) A blasting cap may be defective, due to the fact that water has penetrated the cap or to the fact that the platinum bridge in the cap has become unsoldered; (2) short-circuiting may be caused by a half-hitch taken with the fuse wire around the primer (which is a poor but common practice) which may have broken the insulation so as to permit the electric current to pass from one wire to the other without passing through the cap; but in this case charges in all other holes of the series will explode; (3) a defective splice in the connecting wires may have broken the circuit;(4) a fuse wire may have been broken in the process of tamping; (5) the battery may be overloaded. This last cause is one of the most common causes of misfiring. Any given battery will explode a limited number of caps in series through a given number of feet of copper wire of given diameter. Increase the number of caps, or increase the length of wire, or decrease the diameter of the wire, and the battery will fail to explode the caps. The copper of the leading wires should be at least twice as thick as that of the connecting wires in order to reduce the resistance to the passage of the current as far as possible. Never load a battery up to its limit, but have a good margin of surety that it will explode all the caps in the series. Saunders is authority for the statement that a weak battery may explode part of the caps and leave the rest unexploded, due to variations in the resistance of the platinum bridges in the caps. In case of a misfire no one should approach the holes for half an hour if electric firing is used, and not for several hours if fuse firing is used. This is the rule laid down by several authorities; but it is doubtful whether it is ever followed in practice. I fail to see any good reason for waiting more than a few minutes after a misfire by electricity; but with a fuse there is always danger that the flame may smoulder and creep slowly past some break in the powder thread and finally explode the cap. After waiting some time it may be necessary to remove part of the tamping in the hole and put down another "primer." This is a dangerous operation at best, and if black powder is used a copper or wooden (never steel) spoon should be used in removing the tamping. In any case never remove the tamping entirely, but leave the 3 or 4 ins. of the cushion tamping above the charge in place. Then place several sticks of dynamite and a "primer" on top of the first charge and fire again. The New York City rules forbid removing tamping at all, and require that a new hole shall be drilled not closer than 12 ins. to the old hole, and that in this new hole a heavy charge be loaded and fired. Whenever an explosion fails to carry away the rock clear to the bottom of a drill hole it is forbidden to begin drilling in the bottom of the old drill hole, as part of the former charge may remain unexploded in the bottom of the old hole and explode under the blows of the drill. I question whether it is always safe practice to drill a new hole within a few inches of an old hole, hoping to be able to explode the charge in the old hole by a blast in the new hole. A safer practice, which I have followed in open-cut work, is to drill the new hole several feet from the old hole and to a depth that will bring the bottom of the new hole on a level with the top of the charge in the old hole. Then upon blasting the new hole the shattered rock around the old hole may be removed, the dynamite exposed, a cap inserted and fired. Joveite possesses a decided advantage over any other explosive in common use, when it comes to removing tamping after a misfire due to a poor cap; for there is no danger of exploding it either by shock or by a spark. Care should be taken, however, to use a copper or wooden spoon in removing the last part of the tamping, for while a spark will not explode Joveite, it will set fire to it.

Methods of Firing.—Black powder is exploded by direct contact with any incandescent substance, such as the burning train of powder of a safety fuse; but dynamite, Judson powder and Joveite are exploded only by detonators, or "caps," as they are commonly called; the cap in turn being exploded either by a safety fuse or by an electric current. There is absolutely no advantage in using a cap to explode black powder, for it will not produce any greater effect. Black powder cannot be detonated, but always explodes slowly.

Safety Fuse.—William Bickford, of Cornwall, patented his justly celebrated safety fuse in 1831. It consists of a powder thread around which is spun jute yarn, which is afterward waterproofed with coal tar. The core of powder is so tightly compressed in a thin thread that the fire travels along it slowly, the rate in a good fuse being 1 ft. in 1-3 to ½ min. Single fuses have only one layer of waterproof yarn around the powder; double fuses have two layers of waterproof yarn. Tape fuses are wound with waterproof tape, overlapping. In wet holes double fuse and tape fuse are used. For blasting under water gutta-percha covered fuse is used.

Caps.—A cap (also called a blasting cap, a detonator, or an exploder) for exploding dynamite consists ordinarily of a mixture of mercury fulminate and potassium nitrate or chlorate placed in a small copper capsule, the open end of which is plugged with sulphur, if the cap is one made to be fired with electricity; but if the cap is to be fired with a fuse the fulminate is covered with shellac, collodion, thin copper foil or paper, and the end of the capsule is left open to receive the end of the fuse. Caps for use with a fuse are 1½ ins. long and of 22 caliber.

Caps are commonly graded according to the amount of fulminate in the cap, as follows:

Single strength	(X.)	3	grains of mercury fulminate
Double "	(XX.)	6	" " " "
Treble "	(XXX.)	9	" " " "
Quadruple "	(XXXX.) ..	12	" " " "

And so on.

There is, however, no set rule or law in the United States as in England governing the grading of caps. In England the manufacturers are compelled to grade their caps as follows:

Grade of Cap.......	No. 1	No. 2	No. 3	No. 4	No. 5	No. 6	No. 7	No. 8
Charge Fulminate grs.....	4.6	6.2	8.3	10	12.3	15.4	23.1	30.9

Fulminate of mercury when heated to 367° F., or when forcibly struck, explodes with great violence. The presence of a small per cent. of moisture prevents explosion; hence caps stored in a damp place deteriorate rapidly. Mr. W. J. Orsman has shown conclusively how quickly caps deteriorate in a damp place by placing a few caps in a bottle on top of some damp saw-dust. In 24 hrs. the caps had absorbed 0.1 per cent. moisture; in 14 days, 0.4 per cent.; in 22 days, 0.5 per cent. After 40 hrs. the caps failed to explode dynamite, although they would still explode themselves. Mr. A. W. Warwick has shown * how important it is to use powerful caps in exploding dynamite to get the best results. To test the strength of caps he recommends standing a cap on a sheet of lead 1½ × 1½ ins. × ⅛ to ¼ in. thick, enclosing it with a pipe and exploding it. A strong cap will pulverize the copper shell and the particles of the shell will make fine marks in the lead, around the deep hole left where the cap stood. A weak cap will not pulverize the copper shell, but will tear it into larger pieces, which make large marks in the lead around a shallower hole where the cap stood.

The stronger the cap the more powerful will be the explosion of the dynamite. This is well shown by the following tests made by Mr. Warwick and published in *Mines and Minerals.*

* *Mines and Minerals,* Sept. and Oct., 1902, and Feb., 1904.

The dynamite was tested with the "Abel block." An "Abel block" is a cylindrical piece of lead, 5 ins. diameter by 5 ins. high, with a ¾-in. hole, 2½ ins. deep, in which a charge of 5 grams of dynamite is placed in the form of a small tissue paper enclosed cartridge. The cartridge with its cap is pushed home and tamped with sand that has passed a 60-mesh screen. The block is placed between two $1 \times 6 \times 6$-in. iron plates and the whole held together with two iron rings, $1\frac{1}{2} \times 1\frac{1}{2}$-in. section, and securely wedged. The fuse is fired and the resulting cavity in the lead is measured by pouring in water, then by deducting the volume of the original ¾-in. bore hole, the increased volume due to the explosive is ascertained in cubic centimeters. To determine the theoretical efficiency a simple proportion serves:

35 per cent. powder : 40 per cent. powder : : 129.3 cu. cm. : 141.2 cu. cm.

Careful tests showed not to exceed 4 per cent. variation in the effectiveness of samples taken from different parts of the same commercial stick of dynamite, while even less variation was found in sticks taken from various parts of different boxes. This leads to the very important conclusion that misfires in mining or quarrying cannot be attributed to lack of uniformity of cartridges, when the dynamite is not frozen. Tests on two different makes of dynamite showed the effectiveness of varying percentages of nitroglycerin thus:

30 per cent. dynamite	35.3	35 per cent. dynamite	44.5
40 " " "	43.1	40 " " "	47.6
60 " " "	61.7	60 " " "	63.3

These results show conclusively that the absorbent dope of low-grade powders adds materially to their effectiveness.

Another set of experiments was made to show the relative strength of a given dynamite using different strength of caps:

	3x cap.	4x cap.	5x cap.
35 per cent. dynamite	37.4	40.2	44.7
40 " " "	40.9	41.6	46.8
60 " " "	59.7	62.2	63.3

From which we see that with 35 per cent. dynamite, the 5x cap gives 19½ per cent. increased efficiency as compared with the 3x cap; with 40 per cent. powder, 15 per cent. increased efficiency; 60 per cent. powder, not quite 6 per cent. increased efficiency.

As a further confirmation of these tests the running of three cross-cut drifts in diabase showed similar results. The drilling was done by hand hammers with ⅞-in. drills, and 40 per cent. dynamite was used. The work was done in winter when the outside temperature was many degrees below zero. The different caps were used for one week in each of the three drifts, and the advance carefully measured. The "duty" of the dynamite was measured in cubic feet of rock loosened by a pound of dynamite, and was as follows:

	3x caps.	4x caps.	5x caps.
Drift No. 1	19.4	23.5	22.8
" No. 2	17.6	24.2	25.1
" No. 3	18.7	22.6	23.7
Average "duty"	18.6	23.4	23.8
Cost per ft. of drift	$6.34	$5.75	$5.72

As a result of these tests, 5x caps were used in winter and 4x caps in summer. When the 3x caps were used the dynamite fumes were so bad as to make it impossible for the men to work well. Caps should always be stored in a dry atmosphere; keeping them long underground is apt to weaken them materially.

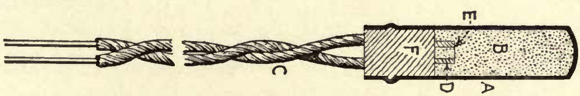


Fig. 12.

Electric Detonators.—The electric detonator or "platinum fuse," as it is called by some makers, has a composition similar to the fuse cap described in the previous paragraph, and all that is there given regarding strength and use of caps applies to the electric cap, or "fuse." Fig. 12 shows an electric cap in which A is the copper shell; B the fulminate of mercury; C the insulated copper fuse wires; D the bare ends of the fuse wires projecting through the sulphur plug, F; E the platinum wire called the "bridge" soldered to the fuse wires. This platinum "bridge" is heated red hot when the electric current passes through the wires, and thus explodes the fulminate of mercury. The fuse wires, C, are 4 ft. to 30 ft. long, depending upon the depth of the blast hole. They are insulated with cotton. The ends of these fuse wires are connected to "connecting wires" that reach from hole to hole. The sulphur plug does not make detonators entirely waterproof, for the copper of the shell expands more than the sulphur upon any rise of temperature in the air, and thus opens a slight crack through which moisture may reach the fulminate of mercury. Shoemakers' wax, warmed, then cooled until jelly like, if daubed around the end of the exploder, will keep out water when using exploders in wet holes. Tallow may also be used, but is not so effective. In nine cases out of ten, failure to explode a series of holes by electricity is due to a defective exploder. The cause may be moisture that has reached the mercury fulminate while the caps were stored or after charging in the hole; or it may be that the platinum bridge has broken from the copper fuse wires. Care should be taken never to pull hard upon the fuse wires, for fear of loosening the platinum bridge.

CHAPTER IX.

METHODS OF BLASTING.

The Theory of Blasting.—The rules commonly found in text books are based upon theories which in turn are founded upon assumptions as to conditions not often encountered in practice. One may safely look with suspicion upon any dogmatic rule as the amount of explosive to use and the spacing of blasting holes. In the first place most of the "rules for blasting" were originated by users of black powder, and are valueless when applied to high power explosives. In the second place, many of the rules apply only to single shots, and not to the simultaneous firing of many holes by electricity. In the third place, practically all of the rules ignore the use to which the blasted rock is to be put. The last is perhaps the most conclusive reason why "rules for blasting," no matter how high the authority back of them, are apt to be exceedingly misleading.

Rock that is quarried for dimension stone masonry requires close spacing of blast holes on a straight line, but these rows of holes themselves may be several feet apart. When holes are drilled close together in this way and fired simultaneously, using very small charges of black powder in each hole, a huge block of solid rock may be wedged off without shattering it.

Rock that is quarried for concrete or macadam purposes should be well shattered to save the cost of sledging and "blockholing" into sizes that will enter the crusher. This means an entirely different spacing of the holes from dimension stone quarrying, and it usually means the use of a high power explosive that will thoroughly shatter the rock.

In the two cases just cited the rock is put to some use after it is quarried; but in open-cut work on canals and railways the rock is frequently wasted, and is broken up only to as small sizes as can be handled conveniently by the appliances available. In some cases these appliances are simply crowbars for levers and boards, for inclined planes, up which the stone may be rolled by hand into wagons. In other cases derricks that can lift only a ton or so are available; while in still other cases cableways that can handle a mass of ten tons are to be used; or it may happen that steam shovels are to load the stone, in which case it must be shattered to comparatively small sizes.

With all these variable factors, how absurd it is to lay down any inflexible "rules for blasting," and yet we find such rules in every text book. Often, it is true, the author forewarns us that judgment must be used in applying the rules, but neglects to tell us afterward where and how to acquire that judgment. Indeed, by the very act of omitting to say anything further as to the exercise of judgment, the author permits us to forget that he has told us that the rules must be used with judgment. I cannot hope to be able to explain how blasting should be done in every case, but I hope to keep constantly before the reader the desirability of reasoning, instead of persuing the much easier, but in the end the more costly, practice of following "rules."

The Crater Theory.—Regarding the theory of blasting, much more has been put into print than is warranted by the meagre scientific experiments made by the writers. A commonly quoted theory of the action of an explosive is one that may be termed "the crater theory." According to this theory an explosive buried in a mass of earth or rock will blow out a funnel-shaped crater whose sides are supposed to have a slope of 1 to 1, if the surface of the earth or rock is horizontal, as shown in Fig. 13. The distance D B; or, to be more exact, the distance D F from the surface of the rock or earth to the center of the charge of powder, E B, is called the "line of least resistance." The volume of the funnel or crater is:

$$V = \tfrac{1}{3}l \times \pi l^2 = l^3 \text{ (nearly)}$$

Hence the general formula for the volume of rock loosened by one charge so as to form a funnel crater is:

$$V = m\, l^3$$

According to Schoen, m = 0.4 for tough, soft rock.
m = 0.9 for hard, brittle rock.

This means that in tough soft rock the sides of the crater are steeper than 1 to 1, but in hard, brittle rock they are nearly 1 to 1.

Note carefully that this theory assumes two things; first, that only one charge of explosive is fired at a time; second, that the rock is homogenous and has no seams or cracks. Except for chamber blasting, this theory, in my opinion, is not worth the ink it is printed with; because, in practice, shots are not fired singly nor is the rock ordinarily free from seams and joints.

ey are!

If the drill holes were bored vertically in rock, as in Fig. 13, and heavily charged with black powder, it might blow out the tamping and fail to rupture the rock at all. The writers who have accepted the "crater theory" have, therefore, reasoned that drill holes should be drilled at an angle with the surface, along some such line as A B, instead of along the line of least resistance, D B, as shown in Fig. 13. If

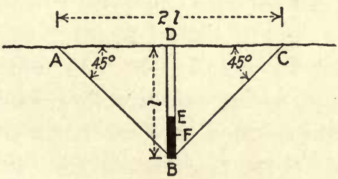


Fig. 13.

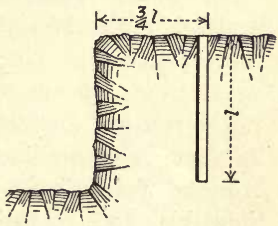


Fig. 14.

this were done the length of the line of "least resistance," D B, would be about 0.7, or nearly ¾ the length of the drill hole A B. Hence we find the rule laid down with the utmost dogmatism, that "the line of least resistance should never be more than three-quarters the length of the drill hole." Then they go a step further and show us a drill hole in a bench, as in Fig. 14, placed back from the face so that the distance from the face is three-quarters the depth of the hole, and they tell us that this ratio should never be exceeded. With single shots, with black powder, with a rock perfectly homogeneous, we may concede that there is some reason in this rule, but as it is given in text books without any qualifications or explanations at all, the rule is worthless. With dynamite I have often placed a row of holes a distance back from the face just twice as much as this rule permits; and the result has been excellent when the rock was much seamed and easily broken by holes fired simultaneously.

The advocates of the crater theory of blasting recommend that the "rock coefficient" be obtained for any given kind of rock as follows: Drill a row of holes 2 ft. back from the vertical face of rock to be blasted down in benches. Let these vertical holes be 6 ft. apart and 3 ft. deep. Charge one hole with a very small charge of explosive; charge the next hole with a greater charge; and so on, charging each hole heavier than the last. Tamp and fire the holes separately, and select that hole which has given the best results—the one that has broken rock without hurling it far. Suppose this particular hole was charged with 0.5 lb. of dynamite, then the "rock coefficient" is:

$$\frac{0.5}{2^3}=\frac{0.5}{8}=0.06$$

The rule is: Divide the charge of the explosive in pounds by the cube of the line of least resistance in feet to get the "rock coefficient." Then, they tell us, we have only to multiply this "rock coefficient" by the cube of the line of least resistance that we intend to use in practice, to ascertain the proper charge for each hole. Thus, if we are going to drill holes so that the line of least resistance will be 8 ft., we multiply 0.06 by 8^3, or $0.06 \times 8 \times 8 \times 8$, and we get 30.72 lbs. as the proper charge per hole. I have tested this rule, and, as I had anticipated, I have found that it gives too large a charge for deep holes in stratified rock with a long line of least resistance.

Wherein is the "crater theory" defective? To begin with it assumes that the work the powder has to do is entirely dependent upon the volume, or weight, of rock to be moved; that if 0.8 lb. of powder will break 1 cu. yd. of rock in a shallow hole, the same proportion of powder will break the rock in precisely the same manner in a deep hole, requiring always the same fraction of a pound of powder per cu. yd. whether the holes are deep or shallow. Every experienced blaster knows that this is not so. Generally the deeper the holes (and consequently the longer the line of least resistance) the less the number of pounds of powder required per cubic yard. We have, therefore, a *reductio ad absurdum* so far as the crater theory is concerned. Moreover, when we consider the work that an explosive has to do, we find no good reason for assuming that equal weights of powder will break equal weights of rock regardless of depth of holes. The force of the powder is expended in four ways: (1) shearing the rock loose; (2) in overcoming the inertia of the rock mass; (3) in heating the rock; and (4) in imparting motion to the surrounding air. If all the work were expended in shearing the rock, then since the volume of a crater varies as the cube of the line of least resistance, while the area of the slopes of the crater varies as the square of the line of least resistance, it is evident that the shorter the line of least resistance the greater the unit work done by the powder in shearing loose the rock. Thus a crater 2 ft. deep, having an apex angle of 90°, has an area of side slopes of 35.4 sq. ft. and a volume of 8.4 cu. ~~yds.~~ ft., or 4.2 sq. ft. per cu. ft. A crater 8 ft. deep has an area of 567 sq. ft. and a volume of 536 cu. ft., or 1.06 sq. ft. per cu. ft. Hence the work of shearing off the rock in the 8-ft. crater is about one-fourth as much per cu. yd. as the corresponding work in the 2-ft. crater. In a well-charged blast, where the rock is merely shattered, but not heaved, the work of overcoming the inertia of the rock is comparatively slight; but we do not know, and can at present not even guess, what portion of the energy of the powder is entirely lost in heating the rock and in imparting motion to the air. For all that can now be proved to the contrary, we might assume that this percentage of lost work is a constant percentage in large and in small blasts. If so, the crater theory crumbles at once into meaningless words; if not so, the advocates of the crater theory still have to prove that their primary assumptions are based upon reasons having even the semblance of scientific truth. I have a great respect for any one who formulates a theory that accords with experience; but I have no respect for a theory that utterly fails to help the practical man and actually misleads him, as exemplified in the crater theory of blasting.

Placing Drill Holes.—In the days of black powder it was essential to consider carefully the position of seams and bedding planes in the rock so as to place the drill hole where the powder would have the least possible work to perform in shearing off the rock. Upon the introduction of dynamite it was found that less care need be taken in placing the drill holes with regard to natural seams in the rock, excepting in quarrying dimension stone. Nevertheless, some attention should always be given to the position of planes of weakness, especially in stratified rock. In open-cut work, where the rock is excavated in benches, it is well to have the bottom of the drill hole stop just short of a plane of stratification or weakness in the rock, even if to do so necessitates drilling holes somewhat shallower than they would be drilled in rock uniformly solid. In tunneling and shaft sinking, where hand drills are used, care is always taken to locate the "cut holes" with reference to seams or planes of weakness in the rock. If the strata in a hand-driven tunnel dip downward toward the face, drill and fire the first holes near the roof; if the strata dip down away from the face, drill and fire the first holes near the floor. In any case drill a hand-driven hole as nearly perpendicular as possible to the planes of weakness in the rock. The spacing of holes will be given in subsequent chapters.

Springing Holes.—In order to enlarge a drill hole so that a greater charge of powder may be placed in the hole, a few sticks of dynamite may be exploded at the bottom of the hole so as to make a chamber there. This process is variously termed "chambering," "springing," "shaking," "bullying," etc. In earth and soft rocks like shale, the dynamite used in springing the hole compresses the material at the bottom of the hole and thus enlarges the hole; in hard rock the dynamite pulverizes some of the rock and hurls the powdered rock out into the air, leaving a chamber. In very soft material the first springing will make a sufficiently large chamber; but in hard rock repeated springing, with increasingly large charges of dynamite, becomes necessary. Thus in springing 20-ft. holes in sandstone I have used for the first springing shot 2 sticks of 40 per cent. dynamite; for the second springing shot, 5 sticks; for the third shot, 20 sticks. The chamber thus made was charged with 8 kegs, or 200 lbs. of black powder.

Springing can be used with great economy of explosives in open-cut work where deep holes can be thus enlarged and charged with black powder or Judson powder. It is indeed surprising to note how often holes are charged with dynamite and fired without any attempt to test the springing method of blasting. On the other hand, I have frequently seen the springing method used under conditions not at all favorable; thus in 6-ft. holes in hard limestone, where a sewer trench was being excavated, the contractor was firing in successive shots a total of 8 or 10 sticks of dynamite for the sake of crowding a few more sticks into the bottom of the hole for the final shot. A much more economic arrangement of powder in sewer trenches is to distribute it in small, separate charges from the bottom to near the top of hole, and not to concentrate it at the bottom. In tunneling, on the other hand, it is always desirable to concentrate as much of the charge as possible at the bottom of the hole; and, were it not for the fumes and dust produced by springing, it would always be good practice in railway tunneling through comparatively soft rock to spring the holes. With water for spraying the air after each shot the objection to springing the holes would disappear. In stoping, shaft sinking and tunneling there is every reason for concentrating the charge as much as possible at the bottom of the hole, and no pains should be spared in experimenting with the springing method of blasting, even to the installation of a water supply system for clearing the air of dust and fumes after each springing shot. In open-cut work it is not so essential to have the charge in the bottom of the hole if dynamite or Joveite is used; but if black powder or Judson powder is to be used a sufficient quantity cannot be charged in the hole of ordinary diameter unless the holes are placed close together. The use of the springing method enables the blaster to place the holes far apart in stratified rocks (and thus reduce the cost of drilling), and to load them with large charges of low-grade powder, thus reducing both the cost of drilling and of explosive.

In springing a deep hole it is customary to tamp the springing shot with a small quantity of sand, no attempt being made to fill the hole with tamping. In shallow holes in mines and tunnels less than a stick of 75 per cent. dynamite, well packed, may be used for the first springing shot, lowering or shoving into the hole a small "primer" containing the cap; tamp with fine sand, and fire.

After the hole has been sprung, if it is to be loaded with black powder, care should be taken not to put the powder in before the rock has cooled off. It sometimes happens that the chamber made by springing is not large enough to hold the necessary charge of powder; in such cases there is an advantage where Judson powder has been used, for the Judson powder may be ignited and will burn up without exploding, thus making it possible to enlarge the hole by further springing. Where the rock strata are inclined at a high angle it sometimes happens that the shock of springing will cause a slip, closing up the chamber and causing a loss of the hole.

Blasting with Powder and Dynamite Together.—In blasting blue sandstone on the Wabash Railroad in eastern Ohio, Mr. W. M. Douglass, of Douglass Bros., has found that the most effective way of blasting so as to reduce the stone to small sizes easily handled by a steam shovel is to fire large charges of black powder and dynamite in alternate rows of holes. * Figs 15 and 16 show two typical blasts, Fig. 16

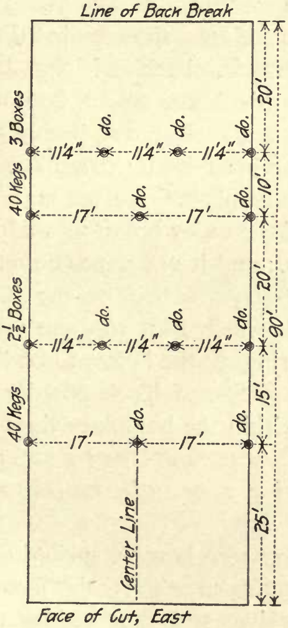


Fig. 15.

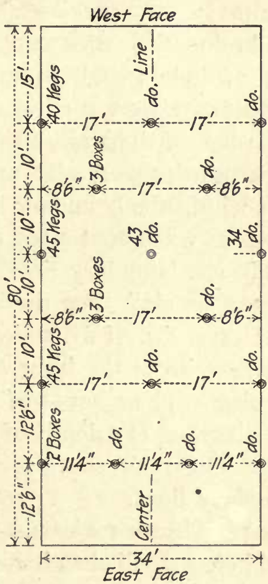


Fig. 16.

being the last blast fired when there were two faces. The holes were all drilled 24 ft. deep with a well driller, the bit being 3 ins. in diameter at the bottom of the hole. The holes

* See page 92 for the cost of drilling in this sandstone.

marked "kegs" were loaded with the number of 25-lb. kegs of black powder given in the diagrams, after springing with dynamite. In springing each hole in Fig. 16, 15 sticks (1¼ x 8-in. size) were first fired, then 40 sticks, then 80 sticks and finally 130 sticks, a total of 265 sticks, or about 132 lbs. of 40 per cent. dynamite per hole for springing. The holes marked "boxes" were loaded straight (without springing) to within 4 ft. of the top, each hole containing as many 50-lb. boxes of 40 per cent. as indicated by the figures in the diagrams. In making a blast the dynamite and black powder were fired together, the theory being that the black powder would lift the rock, while the dynamite would shatter it. The results were excellent. The blast shown in Fig. 15 broke the rock up for 20 ft. back of the last row of holes; and, in making this blast, about 800 lbs. of dynamite were used in springing the six holes, beside the 1,100 lbs. of dynamite used in the blast itself; 6,000 lbs. of black powder were also used in this blast. The amount of rock within the boundary lines of the outer holes, to a depth of 24 ft., was about 2,100 cu. yds.; and it was 2,700 cu. yds. within the boundary lines of the diagram back to the "line of back break." For the blast shown in Fig. 16 there were used 1,600 lbs. of dynamite in springing the holes, 1,000 lbs. of dynamite in the blast and 9,425 lbs. of black powder in the blast. The amount of rock within the boundary lines of the diagram (to depth of 24 ft.) was more than 2,400 cu. yds. The rock was excavated for 2 or 3 ft. outside the boundary lines.

Large Chamber Blasts.—We have seen how by springing a drill hole a small chamber can be made so as to receive a comparatively large charge of explosive, and thus reduce the number of feet of drill hole per cubic yard of rock thrown down. This method may be carried out on a much larger scale by driving a small tunnel or sinking a shaft at the end of which chambers are prepared to receive a great charge of explosive. In this way a mountain of rock may be broken down at one shot, with a great saving in labor and powder. This method of chamber blasting is particularly economic in breaking down banks of hardpan for removal either by steam shovels or by hydraulicking. Unfortunately there is not a great deal in print relative to chamber blasting, but by searching I have found enough to give a good idea of the methods and an approximate idea of the size of charges used. Writers without exception fail to state how much "block holing" was necessary to reduce the chamber-blasted rock to sizes that can be handled by derricks or that will pass through crushers.

The following abstracts of articles on chamber blasting contain valuable information:

In *Engineering News*, May 17, 1900, "W. M." describes "the second largest blast in the history of high explosives," fired Dec. 18, 1899, at West Beaver Creek, Col., by the Pike's Peak Power Co., for the building of a rock-fill dam requir-

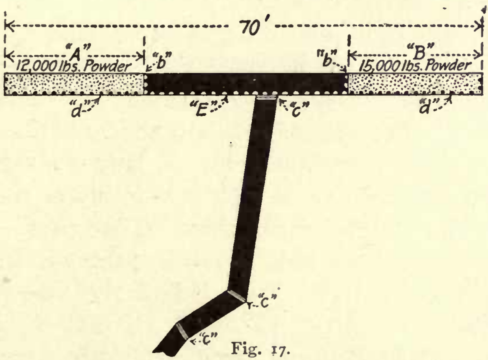


Fig. 17.

ing 42,000 cu. yds. of rock. A granite "butte" or cone of rock was selected and a tunnel run into it 75 ft. below its apex. The tunnel was 135 ft. long, and had several bends in it, Fig. 17, so as to render blowing out of the charge impossible. Cross drifts were run 35 ft. each way from the end of the tunnel. In these cross drifts were charged 32,000 lbs. of black powder and 144 lbs. of dynamite, distributed as shown, and packed solid with bulkheads, b, of sacks of powder. The remaining part of the T was filled with rock and earth, except along one wall where 3,000 lbs. of powder were placed in bags, E. Firing holes, d, 36 in number and 5 ft. deep, containing 4 lbs. of dynamite each, with 3 electric exploders each, were connected in series, making three circuits, so that in case one circuit was found to be open another would be available. The main tunnel was tamped solid with rock and earth, timber bulkheads being placed at c. The firing station was 3,000 ft. away. The explosion opened a crater 72 ft. deep and 150 ft. wide, breaking 110,000 cu. yds. of rock, or 80 per cent. of the rock above the tunnel level. A tunnel through the rim of the crater gave access to the broken rock. Mr. R. M. Jones was the engineer of the company.

Mr. W. R. Russel is authority for the following data on chamber blasting for a rock-fill dam at Otay, Cal.: The rock was a porphyry, easily broken, but very hard to drill. Quarrying was begun by drilling holes by hand 12 to 20 ft. deep, but this was found to be too slow and expensive, so it was decided to run a tunnel and make one large blast. This was done by driving a 4 x 5½-ft. drift 50 ft. and then branching so as to form a Y. The drift was large enough so that double-hand drilling could be used. The ends of the Y were enlarged to form powder chambers. The chamber on the right held 4,000 lbs. of Judson powder; the one on the left held 8,000 lbs. A 50-lb. box of dynamite was placed in each chamber. The drift was completely packed with earth and sand. This blast threw down about 50,000 cu. yds. of rock, at a cost of 3.6 cts. per cu. yd. The cost was:

86-ft. drift	$645
12,000 lbs. Judson	960
Charging	75
Total	$1,680

Part of these 50,000 cu. yds. was further broken up by firing powder in the seams, making a total cost of 5 cts. per cu. yd.

For the second blast a shaft was sunk to a depth of 115 ft. and about 85 ft. back from the quarry face. At a depth of 50 ft. two drifts were run in opposite directions for 25 ft., and a powder chamber was made at each end. At the bottom of the shaft two more drifts were run, one 35 ft., the other 30 ft. The total charge was 15 tons of powder, the greater part being in the bottom chambers. This blast was also very satisfactory. In both cases the electric wires were laid in 1-in. pipes, which were covered by the sand tamping, and the tamping was moistened to make it more effective. After these large blasts there was never any stopping of work to fire, for the larger rocks were block holed and blasted at noon and at night. A derrick delivered the rock to a Lidgerwood cableway of 955 ft. span, capable of handling a 10-ton load. As high as 250 skip loads were handled in 10 hrs., the daily average being 200 loads. The time consumed in hoisting and lowering a skip was 20 per cent. of the time required to make the trip from the quarry to the dam.

A large blast was fired March 3, 1898, by Carpenter Bros. to blow down an isolated mass of trap rock known as "Indian Head," near Fort Washington on the Hudson River. Two tunnels were driven; one near the water edge and 65 ft. deep; the other about 60 ft. from the top and 80 ft. deep. The face was 200 ft. high. Two 25-ft. shafts were sunk from the upper tunnel, and drill holes besides. A charge of 3,000 lbs. of dynamite was placed in one tunnel, and 4,000 lbs. in the other; and it was estimated that with the 7,000 lbs. there were 350,000 tons of trap rock thrown down.

Mr. O. Guttmann, in a paper read before the Inst. of C. E., gives data on chamber blasting on the Danube River. A spur of rock had a vertical face toward the river. A heading 3 ft. wide by 4 ft. high was driven in straight, and then a chamber 6 x 6 x 6 ft. made at right angles to it. The chamber was charged and the heading closed by brick set in cement and by dry stone packing. At first carboazotine was used, consisting of 74 per cent. potassium nitrate, 12 per cent. sulphur, 8 per cent. soot and 6 per cent. bran. It was a low-grade explosive; but, in one blast of 3.9 tons, 25,900 cu. yds. were thrown down where the breast was 60 ft. and the height 99 ft. The largest blast was in May, 1894, when 12 tons of second-grade dynamite (containing 45 per cent. blasting gelatine) in two chambers threw down 3 cu. yds. of rock for each pound of explosive, or practically the same as the carboazotine.

The formula used for charging the chambers was: $L = 3 (v^3 + 5h) q$. In this L is the weight of charge in kilograms*; v the line of latest resistance in meters; h the height in meters of rock above, and q a coefficient depending on the explosive, being 0.22 for carboazotine. The term 5h may be dropped without sensible error. The formula then is almost identical with the formula: $L = 4.19 r^3 c$, used in harbor work at Fiume, where the ratio of height to line of least resistance was kept 3:2. Both these formulas give too high a charge, according to Guttmann.

Mr. J. A. Wilson, in a paper before the Inst. of C. E., gives data on large blasts in New Zealand for harbor works. The stone was granite, gneiss and limestone used in large blocks. On an average 1 lb. of dynamite dislodged 10 tons of stone. Separate charges were proportioned in the ratio of the cube of the least resistance, and this cube of the line of resistance was divided by 35 for dynamite, 36 for gelegnite, 43 for gelatine dynamite, 50 for blasting gelatine and 12 for black powder. Charges of ¼ to 1½ tons were found most effective (a 3-ton charge broke up the rock too much); but this kind of blasting requires a line of least resistance of less than 40 ft. One or more free ends in the quarry with a vertical face are preferable. The length of adit was made

* Kilogram is equal to 2.2 lbs.

nearly half the height overhead, and the chambers were a distance apart equal to 15 times the line of least resistance.

In *Engineering News,* April 2, 1892, large blast firing at Brest, France, is briefly described. Galleries were excavated in the rock and charged with black powder, deposited in barrels covered by boards and tar paper to protect them from seepage water. The galleries were closed for a distance of 13 ft. by stone laid in cement mortar, and then about 7 ft. of dry stone work followed by 7 ft. of stone masonry again. Firing was done by electricity. The amount of powder is not definitely stated, but the author speaks of 40,000 lbs. as being the maximum blast; and in the blast described 104,000 cu. yds. were broken, not a single stone being thrown from the quarry which was in a residential district. At times the ratio was as low as 1 lb. of powder to 11.7 cu. yds. of rock. The rules given below were followed:

(1) The distance between powder chambers should equal the thickness of rock above them.

(2) The face left after a blast should be as nearly vertical as possible to facilitate further work.

(3) With one powder chamber only, the distances from its center to the face of the quarry and to the top of the mass should be equal.

The following data are given in *Engineering News,* Oct. 15, 1881:

"A remarkable feat of railroad building has recently been undertaken from Portland to Dallas, Ore. The road will be 86 miles long. Much of the roadway must be blasted in the flinty face of lofty precipices, or drilled through no less unyielding rock. About 10 miles below Dallas is a bluff of basaltic rock rising 300 ft. from the Columbia River, along whose side the road is to pass. Men suspended by ropes 150 ft. over this wall drill and blast solid rock, their work being attended with the greatest danger. The largest blast on the line thus far has been at a point 10 miles above the Cascades, a mass of rock 165 ft. high, 170 ft. wide and 70 ft. thick at the base, containing more than 40,600 cu. yds. being removed by the explosion of 10,000 lbs. of Judson powder, equal in force to 20,000 lbs. of black powder. The heaviest shot on this work was at 'Jacob's Ladder." At that point 420 cases, or 21,000 lbs., of Judson powder moved 110,000 cu. yds. of solid rock. At 'Shell Rock' 56,000 cu. yds. of solid rock were moved with 10,000 lbs. of Judson powder."

In *Engineering Record* Aug. 10, 1895, Mr. F. A. Mahan tells of large blasts used at Genoa, Italy, in 1895. In limestone quarry, strata dipping 60° toward face, galleries were driven in the base at right angles to each other, and then the supporting pillars were all blown out at once, undermining an area 100 x 300 ft., allowing the strata above to slide down. When the strata were twisted so they would not slide, shafts were sunk from the top. One charge of 11,440 lbs. of dynamite produced a land slide of 260,000 cu. yds. of rock without damaging surrounding dwellings.

A big blast in granite, at Long Cove, Me., is described briefly in the magazine, *Stone* (New York), 1896, p. 555, the data being as follows: A 4 x 4 shaft was sunk 64 ft., then two drifts were run, right and left, each being 27 ft. long; at the end of these drifts cross drifts 26 ft. long were driven, to receive the explosives. Four men were engaged about 8½ mos. doing this work. Black powder was charged in waterproof canvas bags, the men working six days in complete darkness. In each of the two chambers on the west side, 180 kegs were charged; and in each of the two chambers on the east side 185 kegs were charged, making all told 730 kegs of 25 lbs. each, or 18,250 lbs. Thirty-two dynamite sticks, each primed with a cap, were fired to explode the powder. After the explosion the ledge was 50 ft. higher than before (10-ton boulders were hurled 100 ft. vertically) and it was estimated that 1,000,000 tons of granite had been loosened. This appears to be altogether too high an estimate.

Blasting Hardpan.—Hardpan, or cemented gravel, is usually exceedingly difficult to drill because of the "hard heads" or boulders scattered through the mass, and because of the clogging of the drill by the softer material encountered between the boulders or cobblestones. If there are no large boulders in the hardpan, but merely a mass of small pebbles imbedded in clay, or cemented with iron rust, a well-drilling machine can be used to great advantage. The holes drilled by the well driller should be enlarged by springing them with dynamite and then charged with black powder or Judson powder. Dynamite is not effective for breaking down a face of hardpan, because it gives a sudden blow that makes a chamber or pot hole and does not heave the mass of hardpan as does a slower explosive. Joveite, which strikes a blow of great power, but with less suddenness than dynamite, has been used with economy for blasting hardpan for ballast purposes at Cohocton, on the line of the D., L. & W. Railroad. The holes were driven about 7 ft. deep, horizontally into the bank (not vertically), crowbars, post-hole diggers and spoon shovels being used in digging the holes which were about 8 ins. diam. The holes were spaced about 10 ft. apart, charged with 4 lbs. of Joveite in each hole and well tamped. This method of digging horizontal post holes in the face of a gravel bank is one well worthy of remembrance.

On the Chicago Drainage Canal (Chapter XIII.), small tunnels about 2 or 3 ft. in diameter were run into the face of the hardpan. If a large boulder was encountered the tunnel was diverted so as to pass to one side of it. These tunnels were driven at the base of an 18-ft. face, and about 18 ft. center to center.

The late Prof. Thomas Eggleston, of Columbia University, is authority for the following data on bank blasting in California. While the blasting was done for the purpose of breaking up cemented gravel for hydraulicking, it is evident that the same methods are applicable in blasting hard-pan for steam shovel work:

Bankblasting was introduced by J. F. Pierce in 1860, near Smartsville, Cal. Previously the banks had been broken down by undermining with picks, not infrequently burying the laborers. When very hard cemented strata make shaft sinking difficult, then tunnels are driven; but when the bank is not very high, small shafts are usually sunk and enlarged in the form of a bottle at the bottom to receive the powder. When a drift, or tunnel, is driven, the main drift has a drift at its end forming a T. The end drift is about half as long as the main drift. Sometimes a cross drift is run at the middle of the main drift, and has a length about one-third the length of the main drift. The sum of the lengths of all the cross drifts should be about equal to the length of the main drift. The drifts are made as small as the men can work in, generally 3 x 4 ft. The length of the main drift is usually 1 to 1½ times the height of the bank to be blasted, if the bank is a low one; but for very high banks its length is about ¾ the height of the bank. When the bank is 80 to 120 ft. high the main drift is made about as long as the bank is high. For such a drift about 600 (25 lb.) kegs of powder are used, 400 kegs being placed in the cross drifts at the end and 200 kegs in the cross drifts at the middle. In blasting very high banks it has been found wise to run short main drifts connected with cross drifts parallel with the face of the bank; then to charge these cross-drifts and blow out the gravel between the cross drifts and the face of the bank, thus allowing the bank above to fall and break itself by its own weight. In bank blasting with black powder it is generally calculated that 1 lb. of black powder will break 2 to 3 cu. yds. of gravel. It is always better to use too much rather than too little powder, for too little may result in the loss of the entire charge. At the Enterprise Mine 1,700 (25-lb.) kegs were fired at one time in a bank 250 ft. high.

In 1875 a blast of 17,500 lbs. of powder was fired at the

Paragon claim to break a bank 150 ft. high. The drifts had a total length of 325 ft., in the form of a T. The main drift was 110 ft. long, the cross drift on the right side was 70 ft. long and had at its end another 55-ft. drift parallel with the main drift. The cross drift on the left side was 60 ft. long, and at its end had another drift 30 ft. long. The mouth of the main drift was tamped 75 ft. from the end, and the cross drifts were tamped 10 ft. each way from the main drift, which, with the width of the drift, made 100 ft. of tamping. A large amount of open space was left in the L drifts for the expansion of the gases. The electric battery was 450 ft. from the mouth of the tunnel, and the total length of wire was 1,500 ft. The running of the drift cost $300, and the explosives cost $2,700.

A blast of 50,000 lbs. of black powder was fired at the Blue Point Gravel Mine in 1870 that lifted 150,000 cu. yds. (1 lb. to 3 cu. yds.) of gravel vertically a distance of 6 to 10 ft. The main drift (3 x 4 ft.) was 275 ft. long. On the left there were six side drifts, each 120 ft. long, the first one being 75 ft. from the mouth of the main drift. On the right there were six side drifts, each 80 ft. long. On the first drift to the right was a drift 15 ft. long parallel with the main drift. The powder was equally distributed through the drifts, and was fired by electricity from ten different points.

In 1875, at the Dardenelles Mine, 36,000 lbs. of Judson powder broke up 500,000 cu. yds. of cement gravel—a ratio of 1 lb. to 14 cu. yds.—although Judson powder is commonly regarded by bank blasters as being twice as effective as black powder pound for pound. The face of the bank was 175 ft. high and 1,000 ft. long. Into this face five parallel drifts were run, across each of which two or more cross drifts were cut at right angles. The total length of drifts was 1,200 ft. The powder was charged in 28 lots of 1,000 to 2,500 lbs., in each of which were placed three electric exploders.

In 1881 at the Blue Tent Mine 43,000 lbs. of black powder were fired under a bank 200 ft. high.

In 1872, at the Harriman and Taylor claim, 3,500 lbs. of dynamite broke down 200,000 cu. yds. of gravel, a ratio of 1 lb. to 57 cu. yds. This seems to be a very high ratio, for Prof. Eggleston tells of 2,500 lbs. of dynamite loosening 75,000 to 100,000 cu. yds., or 1 lb. to 30 or 40 cu. yds., and adds that 1 lb. of dynamite is as effective as 5 or 6 lbs. of black powder, which, however, does not accord with the data of the blasts cited by him. While dynamite in small charges in drill holes is not effective for bank blasting, as I have had occasion to ascertain by test more than once, it appears to have been very effective when fired in drifts which were undoubtedly not packed solid with tamping, but in which large air spaces were left.

Blasting Piles and Stumps.—In *Engineering News* April 16, 1903, I have given data on pile blasting based upon methods that I used in removing several hundred white oak piles from the bed of the Chemung River, New York. The piles had been sawed off at low water level, when the river was "waist deep." With a ship auger a hole was bored down the heart of the pile a distance of 4 or 5 ft., and then 3 or 4 sticks (½ lb. each) of 40 per cent. dynamite were placed in the hole, tamped with water and fired. The result was merely to splinter the oak into whipcords. After some further experimenting I found that two ½ lb. sticks of 70 per cent. dynamite and one stick of 40 per cent. dynamite in each hole would cut the largest pile off below the bed of the river and hurl the top 50 to 100 ft. up into the air. Ship augers boring a hole 1½ in. in diameter and 4½ ft. deep were used. One laborer bored 7 such holes per 10-hr. day, at a cost of 21 cts. per pile for boring; 1 lb. of 70 per cent. dynamite at 20 cts. a lb., and ½ lb. of 40 per cent. at 15 cts. a lb., made the cost of powder 28 cts. per pile. To this was added 5 ft. of fuse costing 3 cts., and a cap, 1 ct., making a total cost of 55 cts. per pile for boring and explosives. Two men would load and fire 100 holes a day, using water tamping. Once in a while a very tough pile would resist the dynamite, leaving a splintered snag.

Mr. G. W. Stadly, in a letter commenting upon the method of pile blasting, gives the following: It was necessary to remove the piles used for falsework in a river. Rings that would just slip over a pile were made of telegraph wire. On each ring were fastened three half-sticks of 40 per cent. dynamite in separate places, and an electric cap placed in each stick. The rings were dropped over the piles to the bottom of the river, after attaching the electric wires, and, upon firing, each pile was cut off clean, without splintering. It is probable that the piles were of some soft wood like pine or spruce. Eissler states that a tree stump 30 ins. diam. was sawed off at the surface of a river, and three vertical holes (1½ in.) were bored with augers to depths of 4½, 8 and 8½ ft., requiring 3 hrs. for the short hole and 4½ hrs. for each of the others. A charge of 1 lb. of dynamite was fired in each hole.

Stumps may be blasted out either with dynamite or with black powder. A hole is bored in the earth until the end of the hole is directly under the center of the stump. If the soil is clay, an earth auger may be used, or an iron bar may be churned or driven down with hammers. If the soil is dry sand, the hole will fill up as soon as the drill is withdrawn unless some precaution is taken. I have found a neat way to hold the sand is to saturate it with water around the bore hole. This may be done by using a gas pipe for the drill. Drive the pipe down a little way and pour water in at the top, and repeat. If holes are drilled in the sides of the pipe near its lower end the water will run out and saturate the sand, so that when the pipe is withdrawn the hole will remain open. For small stumps half a stick to a stick of 40 per cent. dynamite, well tamped, will serve. In blasting stumps 2 to 8 ft. in diameter, the hole may be sprung (unless in dry sand) by firing a small charge of dynamite, and the chamber so formed loaded with black powder In this way a keg (25 lbs.) of black powder will throw out the stump of a 6 ft. fir tree. Judson powder is very effective for large stumps. In heavy soils place the hole so that the charge of explosive will be close to the roots; but in light soils the powder should be buried deeper. Where the roots spread so as to cover a large area the charge must be placed deeper. In some stumps, such as second growth chestnut, the rotten stump of the first growth lies directly beneath the new tree; in which case the dynamite will merely scatter the old punk and not blow out the new stump. If a large flat stone is shoved under the new stump, between the roots, a stick of dynamite laid upon it and earth packed over it, the explosion will be effective.

Ice Blasting.—To open a channel through solid ice, bore holes through the ice with augers, and suspend dynamite charges of ½ lb. to 5 lbs. each from ½ ft. to 5 ft. under the ice. Solid fresh water ice, 3 ft. thick, has been broken in a circle 60 to 70 ft. in diameter by 4 lbs. of dynamite. Rotten, salt water ice, 10 ins. thick, has been broken in a circle 20 ft. diam. by ½ lb. of dynamite exploded 1½ ft. under the ice. Experiments should be made increasing the size and depth of the charge until the maximum area is broken per pound of dynamite. Charges of 1-3 lb. of dynamite have been fired within 2 ft. of piling, clearing the ice from the piles without damaging them. Ice jams above bridges can often be broken up by firing charges of 5 to 25 lbs. of dynamite. The foregoing data on ice blasting are given in a catalogue on Atlas Powder by the Repauno Chemical Co.

Boulder Blasting.—There are three ways of breaking up a boulder with explosives: (1) Block-holing; (2) mudcapping; and (3) undermining. Block-holing consists in drilling a shallow hole in the boulder and exploding a small charge of high power explosive in the hole. Mud-capping consists simply in firing some dynamite on top of the boulder, after covering it with a shovelful of earth, preferably wet clay. Undermining consists in boring a hole in the earth and firing a charge of dynamite or Joveite in the hole directly beneath the boulder. Block-holing is obviously the most effective way of using the explosive, and it is surprising how small a charge of 75 per cent. dynamite in a block hole will break a huge granite boulder. The cost of drilling is greatly reduced wherever pneumatic plug drills (see page 189) are used. In the Homestake Mine plug drills have largely displaced hand drills for the purpose of block-holing chunks of rock too large to sledge economically.

Mud-capping is very wasteful of powder, and should only be used where a few scattering boulders are to be broken. The "mud" tamping is obviously not sufficient to enable the explosive to do its best work. Undermining is more effective than mud-capping, because the boulder then acts as its own tamping; but very often the earth beneath the boulder is such that boring in it is too expensive, or it may happen that the boulder rests upon rocks. For data on the cost of blasting boulders, see page 232.

CHAPTER X.

COST OF LOADING AND TRANSPORTING.

Cost of Loading by Hand.—Where a laborer has merely to pick up and cast one-man stone into a jaw crusher, I have had men average 34 cu. yds. of loose stone handled per man per 10-hr. shift, which is equivalent to about 20 cu. yds. of solid rock. This, I believe, marks the maximum that may be done, day in and day out, by a good worker, where the stone has scarcely to be lifted off the floor to toss it into the jaws. Every stone, however, was handled and not shoved or slid into the crusher. Going to the other extreme, where conditions are not favorable, where there are more or less delays at blasting, where there is some sledging and a little track laying, and where delays in getting cars are frequent, as in railway tunneling, one man will load about 3 cu. yds. of solid rock per shift (the range being from 2 to 5 cu. yds., as given in Chapter XVI.).

On the Chicago Canal (see page 262) the average output per man per 10-hr. shift was about 7 cu. yds. loaded into dump cars, and this included some sledging. The average per man loading into the low skips used on the cableways, involving very little sledging, was about 10 cu. yds. of solid rock per man per 10-hr. shift. The best day's record was 16.6 cu. yds. per man loading into skips. In loading cars about 5 men out of the force of 36 loaders were kept busy sledging the rock; but with the cableways not only was it easier to roll large rocks into the skips (or "scale pans"), but very large rocks were lifted with grab hooks and chains and carried to the dump without sledging.

In loading wagons with stone easily lifted by one man, the wagon having high sides, I have found that a man will readily average 10 cu. yds. solid, which is equivalent to 17 cu. yds. loose measure per day of 10 hrs. The same man will throw the stone out of the wagon twice as fast as he will load it, and this does not mean dumping the wagon, but handling each stone separately. In loading a wagon having a stone-rack, and no sides, two men, passing stone up to the driver who cords the stone on the rack, will load 1 cu. yd. solid stone in 13 mins. when working rapidly, but this is too high an average to be maintained steadily for a full day. A driver will unload 1 cu. yd. solid (or 1.7 cu. yd. loose) from such a stone-rack, by rolling the stone off, in 7 mins. if he hurries, but he may take 20 mins. if he loafs. A man will readily load a wheelbarrow with stone already sledged and ready for the crusher at the rate of 12 cu. yds. solid (or 21 cu. yds. loose) in 10 hrs.

Croes is authority for the statement that on Boyd's Corner Dam rough rubble stone was loaded onto wagons at the rate of 13 cu. yds. per man in 10 hrs., which is equivalent to 11½ cts. per cu. yd., wages being 15 cts. per hr. The cut stone for this dam, during the years 1868-1869, cost about 30 cts. per cu. yd. to load on stone trucks, but in the year 1870 the cost was reduced to 13 cts. per cu. yd., wages being 15 cts. per hr., although it is not stated how this reduction was effected.

In quarrying mica schist for rough rubble in upper New York City, according to Mr. John J. Hopper the cost of loading wagons was 25 cts. per cu. yd., the rate of wages being 15 cts. per hr.

In moving several hundred yards of stone for rip-rap I have had 5 laborers load, haul 500 ft. on a flat hand car and unload, at the rate of 10 cu. yds. solid measure (17 cu. yds. loose) in 9 hrs. per man. The stone was one and two-man stone, and was handled twice, once in loading and once in unloading.

Cost of Handling Crushed Stone.—In handling stone after it has been crushed to 2½-in. size, or smaller, a shovel is used, and the output of a man depends very largely upon whether he is shoveling stone that lies upon smooth boards or whether it lies upon the ground. I have often had 6 good shovelers unload a canal boat holding 120 cu. yds. loose measure of crushed trap rock (2-in. size) in 9 hrs., but after breaking through to the floor the shoveling was comparatively easy; this is 20 cu. yds. loose (or 12 cu. yds. solid) per man per day shoveled into skips. In shoveling from flat cars into wagons the same rate can be attained, but in shoveling from a hopper-bottom car, where there is at no time a smooth floor along which to force the shovel, an output of 14 cu. yds. loose measure (or 8 cu. yds. solid) is a fair 10-hr. day's work. In shoveling broken stone off the ground into wagons it is not safe to count upon much more than 12 cu. yds. loose measure (or 7 cu. yds. solid) per man per 10 hrs. A careful manager will, if possible, provide a smooth platform, preferably faced with sheet iron, upon which to dump any stone that is to be re-handled by shovelers. Small stone, ¾ in. or less in diameter, is easily penetrated by a shovel and need not be dumped upon a platform. A chamshell bucket operated by a locomotive crane is doubtless the most economic method of loading broken stone from cars or stock piles, where the quantity to be handled warrants the installation.

Cost of Handling with a Derrick.—Where crushed stone must be handled with a derrick, as in unloading boats, I have found the following to be about the best that can be done per day:

	Per day.
6 shovelers, at $1.50	$9.00
1 hooker on	1.50
2 tagmen	3.00
1 dumpman	1.50
1 water boy	1.00
1 team on derrick	3.50
1 foreman	3.00
120 cu. yds. (loose) at 19 cts. =	$22.50

It commonly costs about 25 cts. per cu. yd. (loose measure) to unload a boat of broken stone using skips holding 18 cu. ft. each, and a team on the derrick for raising them. Where any great amount of such work is to be done, however, a hoisting engine and a derrick provided with a bull-wheel should be used. The following, from my note book, shows the cost of unloading flat cars containing broken stone (2-in. size), using a derrick with a bull-wheel for "slewing" the boom:

5 shovelers, at $1.50	$7.50
1 dumpman	1.50
1 engineman	2.50
½ ton coal at $3	1.50
100 cu. yds. (loose) at 13 cts. =	$13.00

In this case a stiff-leg derrick, 40-ft. boom, with a bull-wheel, operated by a double cylinder (7 x 10) engine, handled self-righting steel buckets holding 20 cu. ft. each. Water for the engine was delivered in a pipe. The engineman was the foreman.

In neither of the two cases just cited is the cost of installing the derrick included, nor is the interest and depreciation of plant included. It takes 6 men and a foreman one day to dismantle and move a stiff-leg derrick a short distance (100 or 200 ft.), and one more day to set it up again. This includes moving the engine and the stones used to hold the stiff legs down; and it applies to a slow gang of workmen.

A guy derrick with a 50 or 60-ft. boom swung by a bull-wheel and a hoisting engine will often prove the cheapest device for loading cars with blasted rock. If the derrick is handling skips loaded with stone, the following is a fair average of the time elements in handling each skip load:

Changing from empty to loaded skip	35 secs.
Swinging (half circle)	20 "

Dumping skip	15 secs.
Swing back	20 "
Total	90 "

If there were no delays, it would be possible to handle 400 skip loads in 10 hrs. Usually, however, the loaders will cause more or less delay, so that it is safer to count upon what they will average rather than upon what the derrick can do. One derrick cannot serve a very long face, and the number of men that can be worked to advantage in a given space is always limited; hence I repeat that with a good derrick provided with a bull-wheel the derrick can ordinarily handle more stone than can be delivered to it by the men. The economic size of the skip load is entirely dependent upon the size of the hoisting engine, but a common size skip measures 5 x 6 ft. x 14 ins. deep. Where much work is to be done a contractor should never try to get along with a derrick not provided with a bull-wheel for "slewing" the boom, for the wages of two tagmen would soon pay for a new outfit.

Cost of Loading with Steam Shovels.—A contractor who has never had experience in handling hard rock with steam shovels is almost certain to overestimate the probable output of a shovel loading rock. This is due very largely to the common tendency to think of all rock as being a material that differs only to a moderate degree in hardness. Then again perhaps the frequently published accounts of steam shovels used to load iron ore have had a tendency to mislead the inexperienced man. The iron ore of the Messaba Range, for example, is in reality a material that often requires no blasting and may be dug out of the bank with a powerful steam shovel using a small dipper. It should not be classed as a rock, but rather as a weak shale, so far as cohesive strength is concerned.

A soft shale that can be dug without blasting is just as much a rock as the toughest granite. Yet when it comes to loading rock with a steam shovel there is all the difference imaginable between shales and granites. Practically all printed records of shovels working in rock refer to shale, hardpan and soft iron ore. The only printed records of shovel output in tough rock that blasts out in large chunks are, so far as I know, the records kept by the engineers on one section of the Chicago Drainage Canal. I have given these records in full on page 265. Two 55-ton shovels, each working two 10-hr. shifts a day for four months, averaged 296 cu. yds. of solid rock (limestone) per shovel per shift loaded into cars, although it is stated that one day one of the shovels loaded 600 cu. yds. of rock in 10 hrs. The limestone on the Chicago Canal did not break up into small pieces upon blasting (a condition that is essential to economic steam shovel work in rock), but it came out in large chunks, much of which had to be lifted with chains, instead of being scooped up by the dipper. When each separate rock must be "chained out" in this way, a steam shovel is really no better than a derrick, and is, in fact, not so good.

On a large contract near New York City, where the rock is a tough mica schist that breaks out in large chunks even with close spacing of holes, a 65-ton shovel with a 2¼-cu. yd. dipper averaged for several weeks about 280 cu. yds. of solid rock loaded on to cars. Part of this rock was loaded with the dipper and part was chained. Four men in the pit would fasten a chain around a large rock and throw the hook of the chain over one of the dipper teeth. The shovel would then deliver the rock to the car where a man unhooked the chain. The time required for these four pit men to fasten the chain around a rock ranged from 1-3 min. to 2¼ mins., and on the average was about 1 min. The operation of swinging the dipper both ways and unhooking the rock averaged about 50 secs. Thus a rock every 2 mins. was averaged when working steadily; but delays due to shipping of chains, etc., would bring the average down to about one rock every 2½ mins., or about 240 rocks in 10 hrs. Each rock did not average much to exceed ¾ cu. yd., since the rock broke out in long flat slabs—too long to enter the dipper, although much smaller in cubic contents than the dipper capacity. One of the features that should not be lost sight of in such work is the necessity of close spacing of drill holes in order to break up the rock to sizes such that at least a part of the chunks will enter the dipper. In this case the holes were spaced about 4½ ft. apart. The foreman of this shovel work certainly did not handle rock with the chain as fast as could have been done, for he should have provided an extra chain which the men could have been fastening to another rock while the shovel was unloading into the car.

On the Jerome Park Reservoir excavation in New York City the rock is also a tough mica schist that blasts out in slabs even with heavy blasting. I am informed by Mr. R. C. Hunt, manager for Mr. John B. McDonald, contractor, that their 70-ton shovels loaded only 300 cu. yds. of solid rock per 10-hr. shift. Mr. Hunt says:

> This was in the gneiss rock (mica-schist) of this vicinity. The fibrous nature of Manhattan and adjacent rocks causes it to break in such large and awkwark shapes that the shovel cannot do itself justice. I therefore abandoned the use of shovels in the rock cuts and find that I can handle the rock with derricks more economically.

This statement agrees very closely with my own observation of other contract work on Manhattan, as above recorded. At the times of my visits to the Jerome Park work the holes were being drilled as follows: The face was 35 ft. high, and three rows of vertical holes were put down 25 ft., the rows of holes being 5 ft. apart and the holes in each row 7½ ft. apart. A row of nearly horizontal holes was drilled 35 ft. below the top of the face, the holes being 5 ft. apart. All holes were loaded with dynamite and fired together. The five shovels were loading on to standard gauge flat cars which were unloaded at the dump with a Lidgerwood plow and hoisting engine; the cuts were side cuts.

In thorough cut work on the Wabash Railroad, one 42-

ton shovel loaded 240 cu. yds. of sandstone (solid measure) into dump cars in 10 hrs., as an average of a year's work; but about 10 per cent. of the working time was lost in break-downs, etc.

In shale, or any friable rock that breaks up into pieces that readily enter the dipper, the output of a steam shovel is far greater than in hard rock such as I have been citing. Through the kindness of Mr. George Nauman, assistant engineer, Pennsylvania Railroad, I am able to give the output of several shovels working more than a year, in shale near Enola, Pa. Each shovel worked two 10-hr. shifts per day, six days in the week. In cut No. 1 there were nearly 2,000,000 cu. yds., of which 85 per cent. was rock. Of this rock a little was very hard limestone, some was blue shale nearly as hard, and most of it was red shale, somewhat softer. Excluding the first two months, the average output of each shovel per month of double-shift work was nearly 31,000 cu. yds., equivalent to 15,500 cu. yds. single-shift work. There were, on an average, four shovels at work, averaging 60 tons weight per shovel. The best month's output was 47,300 cu. yds. per shovel in August, 1903, and the poorest month (after work was well started) was 20,800 cu. yds. per shovel in February, 1904, working double shifts. In cut No. 2 there were 1,130,000 cu. yds. of red shale, and while the monthly output per shovel was somewhat less than in cut No. 1, the digging was somewhat better. Three shovels were engaged 13 months, and each averaged 29,500 cu. yds. per month of double-shift work, equivalent to 14,750 cu. yds. of single-shift work. The average weight of each shovel was 60 tons. The best month's work was December, 1903, in which each shovel averaged 41,480 cu. yds. working double-shifts; the poorest month was January, 1904, in which each shovel averaged 23,850 cu. yds. The Allison dump cars used in this work have a capacity of about 4 cu. yds. struck measure; but, although heaped, the average car holds only 2.5 cu. yds. of shale measured in place. The cuts were all side cuts. I spent considerable time in studying the excavation work being done during 1903 between Pittsburg and Philadelphia. Just west of Harrisburg there were 13 steam shovels at work removing some 4,000,000 cu. yds. (mostly shale) for the new gravity yards of the Pennsylvania Railroad. For the most part the cuts were side-hill cuts, and the grades of the temporary tracks were so level that a "dinkey" readily hauled a train of 10 cars, each holding 2.5 cu. yds. of shale, place measure. Each shovel was served by from two to six trains of cars, depending upon the length of haul, and there were few delays in waiting for cars—a vital point in securing economic results. I found that the night shifts loaded about 20 per cent. less material than the day shifts. The crew serving each shovel consisted of 6 pitmen, 1 pit boss, 1 dipperman, 1 craveman, 1 fireman, 3 locomotive engineers, 3 trainmen, 1 switchman, 12 dumpmen and 1 dump boss. There were, besides, about 12 trackmen to each shovel grading new tracks, building temporary trestles, shifting track, etc. Most of the drilling of blast holes, which I have described in Chapter V., was done with well drillers. The shale broke up well upon blasting, often looking like a mass of chips. About 550 cu. yds. of shale loaded per 10-hr. shift was averaged by each 60-ton shovel, including all delays, working in side hill cuts averaging about 24 ft. deep.

I am indebted to Mr. T. S. Bullock, President and General Manager, Sierra Railway Co., of California, for the following data: This company has two 43-ton Marion steam shovels with 1½-yd. dippers. One of these shovels worked from April 1, 1903, to April 1, 1904, in slate rock, all of which had to be blasted. In 300 working days of 10 hrs. each this shovel loaded 199,000 cu. yds. into small horse cars, which is equivalent to 663 cu. yds. per shift. Had large cars been used the output would probably have been 15 to 20 per cent. greater. There were days when 800 to 900 cu. yds. were loaded, and at other times there were delays in waiting for cars, when only 400 or 500 cu. yds. were loaded. This is an excellent record for a year's work.

I am indebted to Mr. Daniel J. Hauer for the following information: With a 65-ton shovel, provided with a rock dipper (shallow and broad) having a capacity of 2½ cu. yds., the output for four months was 15,000 cu. yds. per month, working two 10-hr. shifts per day. The drill holes were 35 to 50 ft. deep, and the rock was granite and gneiss, somewhat disintegrated in places. The drilling was done by hand with churn drills, taking 6 men to pull a drill. The crews were as follows:

6 men, pit crew
8 men, drill crew
1 drill foreman
18 men, dump crew
1 dump foreman
6 men, extra crew
1 foreman.

The "extra crew" at times worked on the dump or helped the drilling crew, and two men were used to run a steam drill (receiving steam from the shovel boiler) in drilling block holes. The cost of repairs to the shovel was very high. The total cost for wages (double shift work), supplies, explosives, etc., was about $8,500 per month. The large number of men on the dump was due to the fact that the rock was all rehandled in widening a fill.

Cost of Steam Shovel Work in Iron Ore.—I am indebted to Mr. Daniel King, General Manager Pinkney Mining Co., Pinkney, Tenn., for the following data: Two 43-ton shovels (1½-yd. dippers), are worked in side cuts 25 ft. high. The ground is generally shaken up in front of the shovel, a 20-ft. hole, 12 to 16 ft. back of the face, being sprung with 2 to 5 sticks of dynamite and then shot with 2 to 4 kegs of powder. The ore, which varies from the consistency of hard pan to ordinary earth, is merely shaken up and not thrown down. The dump cars hold 77 cu. ft. level full and are generally heaped full with dippers, so a car holds not less than 3 cu. yds. of loose material. Each shovel is served by two dinkey engines hauling trains of six cars. The wages paid per 10-hr. day are low, being as follows: 1 shovel engineer, $3; 1 craneman, $2; 1 fireman, $1.25; 4 pit laborers, $1 each; 2 dinkey engineers, $1.75 each; 1 superintendent (to two shovels), $5. There are no firemen or brakemen on the trains. The grade from the shovels to the dump is about 2 per cent. in favor of the load. The following was the cost of operating two shovels one month in 1903. Shovel No. 1 worked 171 hrs.; No. 2 worked 161 hrs.:

No. 1. Excav. 5,513 carloads, wages	$287.65
Transporting 5,513 carloads, 500 ft. haul (one way), wages	83.60
Drilling 272 ft. at 4 cts.	10.88
Explosives	27.52
No. 2. Excav. 3,362 carloads, wages	$243.01
Transporting 3,362 carloads, 1,500 ft. haul (one way), wages	60.13
Drilling 239 ft. at 4 cts.	9.56
Explosives	27.80
Dumping, 8,875 carloads	142.27
Track work	267.60
Trackwork (nights)	75.87
Blacksmith work	76.30
Repair "	110.23
Carpenter shop	37.70
Renewal and repair supplies	196.39
Coal for shovel No. 1 at $4.50 ton	77.30
" " " No. 2 " " "	72.70
Coal for 4 dinkeys	91.00
Iron	23.85
Lumber	37.39
Oil	25.27
Waste	.90
Total for 8,875 carloads	$1,984.92

When washed, these 8,875 carloads yielded 7,633 tons of ore.

In 1902 during a month of 201 hrs. the two shovels dug 10,703 carloads, at the following cost:

Excavating, wages	$533.41
Hauling, wages	250.40
Drilling, 812 ft., at 4 cts.	32.48
Explosives	97.37
Dumping	194.99
Blacksmith work	84.11
Carpenter "	56.40
Repair "	137.28
Track "	185.10
Extra (night) labor	73.30
Renewal and repair supplies	182.46
Fuel	247.40
Iron, $24.60; lumber, $29.23	53.83
Oil, $28.40; waste, $2.28	30.68
Total for 10,703 carloads	$2,159.21

When washed these 10,703 carloads yielded 7,573 tons of ore.

Wages of shovel crew are: 1 shovel engineer, $3; 1 craneman, $2; 1 fireman, $1.25; 4 laborers at $1; 2 dinkey engineers at $1.75; ½ superintendant at $5. There were no firemen nor brakemen on the trains. Grade from shovel to dump, 2 per cent. in favor of loaded trains.

The best day's work with one shovel was Jan. 30, 1903. The work was in a side-cut, 25 ft. high and cost as follows:

Excavating (441 carloads), wages	$12.75
Hauling (400 ft. one way), wages	3.50
Drilling 23 ft. at 4 cts.	.92
Explosives	1.66
Dumping	4.50
Trackwork	18.00
Blacksmith work	3.08

Repair work	5.15
Carpenter work	3.00
Coal at $4.50 ton	5.45
Iron and lumber	3.00
Waste	.83
Total for 441 carloads	$61.84

When washed these 441 carloads yielded 450 tons of ore.

For the sake of comparison the following hand work in this ore will be interesting:

Loaders received 8 cts. per cu. yd. and earned $1.25 to $2 a day; common laborers received 10 cts. per hr. In a month of 235 working hours, with a haul of 800 ft. (one way), 600 ft. being a mule haul and 200 ft. by gravity, in a material not as hard as hardpan, and working to a face 16 ft. high, the cost was as follows:

Digging 2,272 carloads = 4,544 cu. yds.	$363.52
Hauling " " , 800 ft. one way	67.89
Drilling 273 ft. at 4 cts.	10.92
Explosives	36.67
Dumping	32.59
Repair and trackwork	40.68
Blacksmith work	3.10
Iron, $2; lumber, $2.62	4.62
Oil, $2.75; waste, $0.18	2.93
Foreman (1 mo.)	62.50
Total for 2,272 carloads	$625.42

When washed, these 2,272 carloads yielded 2,488 tons of ore. These cars were smaller than the cars hauled by the engines, and they held 2 cu. yds. of loose material.

Cost by Wheelbarrows.—A wheelbarrow load averages about $^1/_{25}$ cu. yd. of solid rock. A man will load such a barrow in 2 mins., and will walk with it at a speed of 180 ft. per min. if he is lazy and to 250 ft. per min. if he is active; and he will lose ¾ min. each trip in dumping the barrow, fixing run planks, etc. Assuming a speed of 200 ft. per min. and wages 15 cts. per hour, the cost of loading, hauling and dumping is:

Rule I. To a fixed cost of 18 cts. per cu. yd. of solid rock add 6¼ cts. per cu. yd. per 100 ft. of one-way haul from pit to dump.*

Cost of Hauling in Carts and Wagons.—Since a cubic yard of loose, broken stone weighs about as much as a cubic yard of earth measured in place; and since, ordinarily, 1 cu. yd. of solid rock becomes 1.7 cu. yds. when broken, we may say that a team will haul about 60 per cent. as many cubic yards of solid rock per day as of earth. In other words, if the roads are such that 1 cu. yd. of packed (not loose) earth would make a fair wagon load for two horses, then 0.6 cu. yd. of solid rock would be a fair load. In my book on earthwork I have discussed in considerable detail the sizes of loads of earth that teams can haul, and it is only necessary to multiply the earth load as given there by $^6/_{10}$ (or 60 per cent.) to find the equivalent load of solid rock. Another way to estimate loads is to use the ton of 2,000 lbs. as the unit. Solid rock seldom weighs more than 2.2 tons per cu. yd. Over poor earth roads, with occasional steep pitches, a load of 1 ton is practically all that an ordinary team should be counted upon to haul, or less than ½ cu. yd. of solid rock. If the road is hard and level, a team will haul 1 cu. yd. of solid rock; or one horse will haul ½ cu. yd. in a cart. If the road is a good macadam all the way, with no grades over 4 per cent., and no pulls through soft earth, a good team can haul about 1½ cu. yds. of solid rock, but these conditions are exceptional. In ordinary city and village work, and on level hauls over hard earth roads, assume 1 cu. yd. of solid rock as a load for two horses. A team travels

* In this rule, and in the rules that follow, I have given not the lowest records of cost that I have, as will be seen by anyone who takes the pains to study this book well. I have preferred to give conservative estimates of cost in all the rules. It is never safe to estimate too closely before work has been actually begun; but once it is under way every cent per yard should be looked after with the greatest diligence. In a word, make your saving in your work and not in your preliminary estimate.

220 ft. per min., or 2½ miles an hour, at a walk over ordinary earth roads, a little faster over good pavements and a little slower over soft roads, the variations from this average of 2½ miles an hour being seldom more than 20 per cent., making it about 2 miles an hour over poor roads to 3 miles an hour over the best macadamized roads. It is perfectly safe to say that a team can walk steadily for 8 hrs., averaging the speeds above given, going loaded and returning empty; so if the shift is 10 hrs. long, and not over 2 hrs. are lost in loading and dumping, the team has 8 hrs. to travel, in which time it will cover 16 miles over poor earth roads, 24 miles over good macadamized roads and 20 miles over ordinary earth roads. If the hauls are short it may happen that so much time is lost in loading and dumping that the team has considerably less than 8 hrs. of actual walking time left. Each case must be considered by the contractor.

As to the wagons used for hauling one and two-man stone, my own preference is for an ordinary wagon from which the box has been removed and replaced by a "stone rack." A stone rack is 3 ft. wide and 11 ft. long, its floor being 3-in. plank and its sides and ends nothing but 3 x 4-in. strips. This makes a "box" that is low and easily loaded, when necessary big stones being rolled up an inclined plank onto the wagon. It is also unloaded easily, large stones being simply rolled off without lifting. Where hauls are very short, and the stone all broken to one-man size, a patent dump wagon may be used advantageously; but such a wagon always weighs much more than the common wagon with a stone rack, aside from the fact that a patent dump wagon is always harder to load. As above stated, a driver will unload 1 cu. yd. solid measure (or 1.7 cu. yds. loose measure) of stone from a stone rack in 7 mins. if he is vigorous in his work. Certainly two men should never take more than 7 mins. Two men and the driver can readily load 1 cu. yd. onto a stone rack in 15 mins., no stone being heavier than two men can lift. Then if one man and the driver unload in 7 mins., we have 22 mins. team time lost in loading and unloading, which is equivalent to 12 cts. per cu. yd. (team and driver being worth 35 cts. an hour), 8 cts. being for lost team time loading and 4 cts. for lost time dumping. If, including rests, each laborer (exclusive of the driver), averages 7½ cu. yds. loaded in 10 hrs., and wages are 15 cts. an hour, we have 20 cts. per cu. yd. for loading. If one man, assisted by the driver of each team, does the unloading, the cost of his help need not exceed 3 cts. per cu. yd. We have, therefore, a total fixed cost of, 8 cts. lost team and driver time loading, plus 4 cts. ditto unloading, plus 20 cts. for labor loading, plus 3 cts. for helper unloading, making a total fixed cost of 35 cts. per cu. yd. of solid rock. Our rule for loading and hauling and dumping 1 cu. yd. of solid rock (2.2 tons) in wagons, under the above conditions, is:

Rule II. To a fixed cost of 35 cts. for lost team time, and labor of loading and dumping, add $^6/_{10}$ ct. per cu. yd. per 100 ft. of haul (one way) from quarry to dump, or 32 cts. per mile one way; but if the roads are such that ½ cu. yd. of solid rock (1.1 tons) makes a load, to the 35 cts. fixed cost add 1 $^2/_{10}$ cts. per cu. yd. per 100 ft. of one-way haul, or 64 cts. per mile. By using extra wagons, as should be done where the haul is so short that a team cannot be kept on the walk 8 hrs.; or by using more men loading and unloading, as should be done when the hauls are very short, the fixed cost can be reduced to 30 cts. per cu. yd.

In railroad work one driver usually attends to two one-horse dump carts; and as rock cuts are usually higher than the dump the haul is down hill, so that, considering the rough roads, big loads can be handled. By having 4 or 5 men to load each cart, there is about the same amount of lost team time per cu. yd. of rock as above assumed for loading wagons, or 8 cts. per cu. yd.; the cost of dumping is about 2 cts., making a fixed cost for the two horses and driver of 10 cts. per cu. yd., to which add 20 cts. for loading to get the total fixed cost for labor and teaming. With wages at 15 cts. per hr. for laborers and 35 cts. for a driver and two one-horse carts, the rule for loading and hauling by carts, ¼ cu. yd. of solid rock per cart, is:

Rule III. To a fixed cost of 30 cts. per cu. yd. of solid rock add 1 $^2/_{10}$ cts. per 100 ft. of one-way haul. Mr. Daniel Hauer states (see page 226) that he has found 1/3 cu. yd. of solid rock to be a fair average of the size of one-horse cart load on railroad work, but my own records show ¼ cu. yd. to have been an average, and I have preferred to err, if at all, on the conservative side.

Where carts or wagons must be hauled up a steep, bad road, it will often pay to lay either a plank road, or to lay steel channel beams so as to form a trackway. This last method has been used with advantage where a hoisting engine was placed at the top of a long hill to relieve the teams of the work of hill climbing. A boy on a horse can readily drag the snatch rope back down the hill. In the far West it is customary for three or more teams to be hitched to a train of two or more wagons, and, when a steep hill is to be ascended, only one wagon is hauled up at a time. On long hauls this method could be used to advantage much oftener than it is in the East. Snatch teams are not used as often as they should be to enable large loads to be handled over bad spots in the road. Where the roads are fairly good all the year, traction engines are economic.

For hauling cut stone in large blocks, "stone trucks" are used. A stone truck is a strong wagon provided with a platform which hangs below the hubs of the wheels, instead of above them, as in the ordinary wagon.

Where large derricks are available at both ends of the haul, wagon boxes can be made so as to be lifted off by the derricks, both for loading and dumping the rock.

Hauling on Stone-Boats.—For moving large stones a short distance stone-boats or sleds are often used. A stone-boat is a flat platform, ordinarily about 2½ x 4 ft., on wooden runners shod with iron. It possesses three advantages over a wheeled vehicle: First, it is so low that a large rock can be rolled by hand, or dragged by the team on to it; second, it cuts no ruts into wet ground, and, third, it can be dragged about in narrow places. Obviously a team cannot haul a very large load very far on a stone-boat, but surprisingly large loads can be hauled a short distance if the team has long rests between loads. A team of horses weighing 2,400 lbs. can exert a pull of about 1,000 lbs. for a short time if they have a good earth foothold. The sliding friction of iron or wood on earth is about 50 per cent. the weight of the load that is being dragged; hence a team is capable of dragging a stone-boat and load together weighing 2,000 lbs. A team doing such heavy work could probably not average more than 2 hrs. of actual pulling per day. In stone-boat work, however, a stone weighing more than 1,100 lbs. (¼ cu. yd. solid) is seldom handled. Where many such stones are to be hauled a considerable distance, as in boulder quarrying, I have found it an excellent plan to build "skid roads." A skid road is really a rough railroad without the rails, for it is made by partly bedding in the ground round sticks of unsawed timber, like ties for a railway track, 3 to 6 ft. apart. A stone boat with wooden runners, 8 to 12 ft. long, can be "skidded" or hauled along over these ties with surprising ease if the ties are kept well greased. Indeed, a team can thus pull a bigger load than with a wagon wherever there is not a well made road. Where growing timber is at hand a skid road may be made at less cost than grading a wagon road, and it possesses the inestimable value of being a good road even in wet weather. I have seen wagons that were dragged with difficulty through the mud when the load was less than ⅛ cu. yd. of solid rock (550 lbs.); and it often happens in the fall and spring of the year that ¼ cu. yd. of solid rock is a big load for wagons traveling over earth roads badly rutted and muddy. In such cases a skid road can often be built to advantage.

Cost by Dump Cars.—For a discussion of the tractive power of horses and the rolling resistance of cars, the reader is referred to my book on earthwork. On a level track a team will readily haul two dump cars loaded with 3 cu. yds. of solid rock over the ordinary narrow-gauge track with light rails. In railroad work the grade is usually in favor of the load from cut to fill, and it is safe to assume that one horse will haul two dump cars (1½-yd. body) loaded with ½ cu. yd. solid rock in each car; with a well kept track slightly in favor of the load, but not so steep as to stall the horse returning with the empty cars, it is safe to count upon ¾ cu. yd. in each of the two cars.

In a thorough cut the track is usually laid Y-fashion, the two branches of the Y being carried up close to the rock face. Two empty cars are left on one branch of the Y to be loaded while the two loaded cars are hauled away from the other branch. If the haul is short and only a few loaders are at work, only one car is hauled at a time. If the cut is wide enough, and it often is, I prefer to lay two parallel tracks and have two Y's, for in that way the loaders need not take so many steps to get a stone into a car, since there are four places at the face where cars may stand, instead of two. In estimating the cost of loading and hauling in cars, using horses or mules, assuming the rock to be broken up into sizes that one or two men can lift, it is never safe to count upon more than 7½ cu. yds. solid rock loaded per man in 10 hrs., and often it will be wise to estimate on not more than 6 cu. yds. With wages at 15 cts. per hr., the loading costs 20 to 25 cts. per cu. yd. Assuming 1 cu. yd. of solid rock as a fair load for one horse to haul in cars; assuming 4 mins. lost time in changing from the empty to the loaded cars and in dumping; assuming a speed of 200 ft. per min., and assuming wages of driver and one horse at 25 cts. per hr., we have 1 $^{6}/_{10}$ cts. per cu. yd. chargeable to lost time at pit and dump, plus ½ ct. per cu. yd. per 100 ft. of haul (measured one way). The cost of dumping is largely a matter of how many yards are delivered per day at the dump. However, with wages at 15 cts. an hour, the cost of dumping is seldom less than 2 cts., and it may run as high as 5 cts. per cu. yd. Assuming 5 cts. as a fair average of the combined cost of dumping and lost team time, and 20 cts. as the cost of loading by hand, we have:

Rule III. For loading and hauling with dump cars, to a fixed cost of 25 cts. per cu. yd. of solid rock, add ½ ct. per cu. yd. per 100 ft. of one-way haul.

If the distance is great enough to warrant the use of a team of horses instead of one horse, the cost of hauling will be $^1/_3$ ct. per cu. yd. per 100 ft., if wages of team and driver are 35 cts. per cu. yd. I estimate the cost of track laying at $100 per mile, of wear and tear on ties at another $100, and of pulling up track at $50, making a total of $250 per mile, or $5 per 100 ft. of track. Dump cars can be estimated as costing about $50 each, 30-lb. rails at $30 a ton and ties (6 x 6 ins. x 5 ft.) at 25 cts. each.

CHAPTER XI.

QUARRYING STONE.

General Considerations.—In this chapter the quarrying of stone for masonry (other than concrete) will be discussed. Stone that is quarried and split with plug and feathers, or otherwise, to dimensions (ready for stone cutters to begin dressing the surface) is called dimension stone. If it is quarried out in rough slabs or blocks of irregular dimensions it is called rubble stone, or backing stone.

In quarrying dimension stone the first step is to secure a working face in the quarry; the next step is usually to cut or blast a channel at each end of this face, so as to expose three free faces. Then it is possible, by wedging or blasting, to loosen a long block of stone which can be split into short blocks that can be handled by derricks. Where there is a good market for rubble stone, it is not customary to make end channels, but merely to shake up the rock for a short distance back of the face by light blasts, and, if it is a sedimentary rock, large irregular slabs can be barred and wedged out. These slabs can then be squared up by sledging or by plug and feathering, or both. Obviously this method produces a very considerable amount of rubble stone, but it is the common method in small dimension stone quarries.

While in the first chapter attention was called to the joints that exist in stone, it is well to add certain facts to those already given, for the art of quarrying is largely the art of taking advantage of joints and natural cleavage planes. Granite, which to the ordinary eye appears massive and without planes of natural cleavage, has in fact a "rift" clearly seen by the trained eye. Along this "rift" it may be split with comparative ease. At right angles or perpendicular to the "rift" in one direction are planes of cleavage, called the "grain," along which the stone splits with less ease; while at right angles to the "rift" in the other direction are planes of natural cleavage, called the "head," along which it is still possible to split the stone, but with less ease than along the "grain" or along the "rift." These three planes of cleavage are shown in Fig. 18. All sedimentary rocks have a "rift" which corresponds with the planes of stratification or beds; but the trap rocks, like diabase, diorite, porphyry, etc., often have no rift at all, and are consequently unfit for use as dimension stone, since when hammered or wedged they are apt to split, like glass, irregularly.

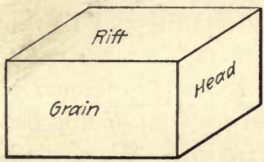


Fig. 18.

The cost of quarrying stratified rocks, like sandstone and limestone, depends largely upon two factors: First, the thickness of the beds, and, second, the "dip" of the beds. The "dip" is the angle, or slope, that the bed makes with a horizontal plane. If the beds lie horizontally, just as when they were originally deposited in the primæval sea, the stone is quarried out in successive layers; and, as these layers usually vary in thickness, the quarryman, after the quarry has been well opened, can select a layer of thickness to suit the demand of any particular purchaser. If the beds dip at a steep slope into the earth, the quarryman must usually remove thick-bedded and thin-bedded stone, all together, as he goes down; and besides he must abandon his quarry, or resort to mining methods, before a very great depth has been reached, because it will not pay to remove the increasingly large amount of stripping. On the other hand, where the beds dip at a high angle the quarryman can determine, by examining the exposed outcrop, what the thickness of each bed is, and can count with some certainty upon the character of each bed. Where the beds lie flat, the thin beds are usually on top, and thicker beds exist below; but this is not always the case, and to determine the character of the deposit diamond drill cores should be obtained.

If the beds of stratified rock are quite thin, the stone may be fit for flagstone, curbing, lintels, paving blocks, slope wall stone, basement masonry and the like; but will of course be valueless for heavy, architectural masonry or for engineering masonry where specifications call for thick courses. This simple fact is frequently overlooked by engineers in drawing masonry specifications, and they often call for thicknesses of courses that either are not to be found in local quarries at all, or, if found, are quarried only at great expense by first removing a lot of thin bedded stone overlying the thicker beds required. Times without number I have seen evidences of such ignorance of quarry products and quarrying costs that obvious as these facts seem I deem them worthy of emphasis. If the beds of stratified rock are thicker than the courses of masonry specified, then the quarried blocks must be split at a cost that should never be overlooked by the quarryman or the contractor in estimating a fair price for his product. Thus it appears that there is a happy medium as regards the economic thickness of beds.

Joints.—All rocks, whether igneous or sedimentary, contain "joints," or seams that run through the natural beds. Often these "joints" are clearly visible, but at times the split may be so thin as to be invisible except to the expert eye. In stratified rock the "joints" as a rule are perpendicular to the planes of bedding, and are spaced quite regularly, as if a giant quarryman had struck the rock with a sledge at intervals, cracking it in vertical planes. The joints in stratified rock have, as a rule, two dominant trends, one set of joints being parallel with the "dip" ("dip or end joints") and the other set at right angles, or parallel with the "strike" ("strike or back joints").

In granites and traps the "joints" occur at irregular intervals and often intersect at varying angles; nevertheless there are generally two sets of vertical joints intersecting approximately at right angles, and frequently there is a third set of horizontal or "bottom joints." If the joints are close together it will, of course, be impossible to quarry building blocks; though, on the other hand, the quarrying of stone to be crushed for concrete or macadam is greatly facilitated by numerous joints, as exemplified in the trap rocks of the Hudson River. Joints are usually quite conspicuous near the surface, due to the fact that changes of temperature have opened them, and solutions of iron salts passing through the joints have stained the rock. But wherever granite is found with numerous close joints in the surface beds, it may be inferred that similar joints exist in the lower beds even if they are invisible, and even if the blocks quarried from the lower beds appear solid. Merrill cites a granite quarry in which the stone at a depth of 25 ft. appeared to be perfectly solid, although above it was full of joints; but upon polished blocks he was able to discover fine hairline joints which eventually would doubtless open up upon exposure. I would suggest that tests on the tensile strength of diamond drill cores would quickly prove the existence or non-existence of such joints.

Where joints in granite run verically and at right angles to one another, as well as horizontally, the quarry is known as a "block quarry." Where there are practically no vertical joints, but where a series of nearly horizontal joints divides the granite into sheets or beds, the quarry is a "sheet quarry." The beds in sheet quarries are usually lenticular in shape, thin at the edges and thick in the middle. In such a quarry blocks 10 ft. thick and 300 ft. long have been loosened.

Plug and Feathers.—Before studying the methods of quarrying it is necessary to understand certain of the commoner tools and machines. Among these the most important are the plug and feathers, shown in Fig. 19. These simple tools are used for splitting large blocks of stone into smaller blocks and for squaring up irregular stones. The plug is the wedge, and the feathers are merely two short pieces of half-round iron whose curved sides fit the sides of the drill hole, while their flat sides receive the thrust of

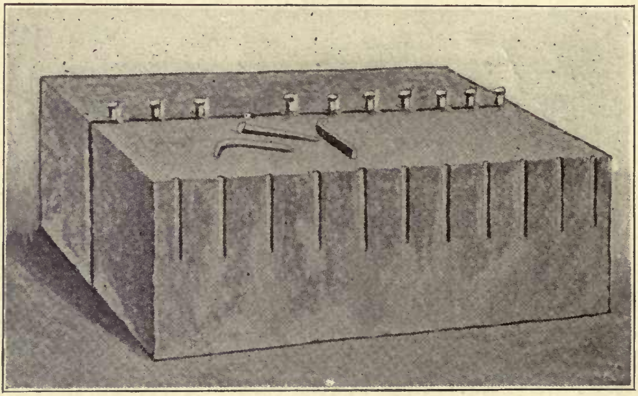


Fig. 19

the plug. It is astonishing to see how thick a block of granite may be split with so small and simple a device. To split a block of granite, a row of holes about ⅝ or ¾ in. diam. and 2½ to 5 ins. deep are drilled about 6 to 8 ins. apart. Then a pair of feathers and a plug are placed in each hole, the plugs being driven home with light blows of a hammer until all are tight. Then each plug in succession is struck one or two blows, the quarryman telling by the ring of the metal under the blow whether the strain is practically the same in each wedge. With plug holes only 5 ins. deep a block of granite 6 ft. thick can be split, leaving a face almost as flat as a board. For granite blocks 3 ft. thick, a hole 2½ or 3 ins. deep will suffice. Some limestones also break remarkably well with shallow plug holes, but marbles and sandstones as a rule require deep holes, although with some sandstones holes 1⅛ ins. x 8 ins. will break a sheet 4 ft. thick, perfectly true, according to Saunders. In most sandstones, however, the holes are usually 1¼ to 2 ins. diam., and, as a general rule, of a depth equal to $^2/_3$ the thickness of the stone. The holes are spaced 4 to 16 ins. apart. In some sandstones the plug holes must be drilled entirely through the stone to insure a true break. The plugs need not always be of the same length as the depth of the hole. Sometimes it is found desirable to alternate deep and shallow plug holes in the same row. In this case the lower half of the deep holes may be drilled with a smaller bit, so that the plugs in these holes will strain only the bottom half of the stone.

For drilling plug holes there are three methods in common use: (1) Drilling by hand; (2) drilling with a pneumatic hammer, called a pneumatic plug drill; and (3) drilling with an ordinary rock drill mounted on a quarry bar. Since plug holes in granite are seldom more than 6 ins. deep (usually 3 ins.), either hand drilling or pneumatic plug drilling should be used; but where deeper holes must be put down a rock drill on a quarry bar should be used.

Cost of Plug Drilling by Hand.—By timing a number of masons at work splitting granite blocks 24 to 30 ins. thick, I found that each man drilled each hole (⅝-in. diam. x 2½ ins. deep) in a trifle less than 5 mins., by striking about 200 blows; and it took about 1 min. for placing and striking each set of plug and feathers. Blocks 30 ins. long, with four plug holes, were drilled and split with the plugs and feathers in 24 mins., on an average. At this rate, a good workman can drill and plug 80 holes in 8 hours.

Cost of Pneumatic Plug Drilling.—For drilling plug holes in granite certainly no tool is as economic as the pneumatic plug drill. Fig. 20 shows one of these drills at work on the Wachusett Dam granite quarry. It will be seen that horizontal as well as vertical holes can be drilled rapidly, which in itself is a distinct advantage over the quarry bar method. The plug drill shown in Fig. 20 does not rotate the bit automatically, which I consider a positive advantage since it simplifies the mechanism and reduces the wearing parts. The operator turns the bit with a wrench, which is such light work as to add little to his expenditure of energy.

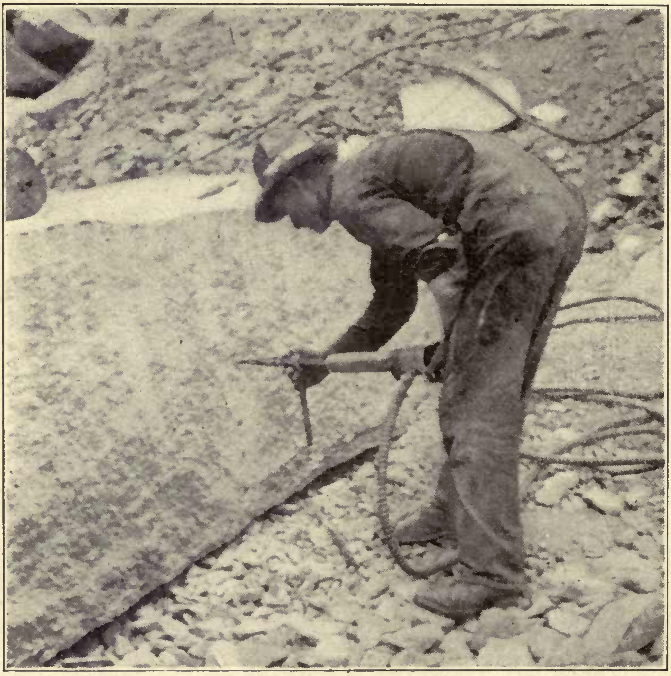


Fig. 20.

The ordinary plug drill, according to the manufacturers, consumes 15 cu. ft. of free air per min. at 70 lbs. pressure. At the Wachusett Dam I found that a workman averaged one hole (⅝-in. diam. x 3 ins.) drilled in 1½ mins., including the time of shifting from hole to hole, but not including the time of driving the plugs. About 250 plug holes are counted a fair day's work for a plug drill where the driller does not drive the plugs himself.

Not only for plug hole drilling, but for block-holing, has the pneumatic drill a promising future. A portable, gasoline-driven air compressor, such as is commonly used for running pneumatic riveters, would serve admirably for plug drilling purposes in quarries where a large compressed air plant is not already installed.

The Quarry Bar.—A quarry bar is a long bar mounted on four legs, and upon the bar the drill is mounted, so that the drill can be moved quickly from hole to hole along the bar,

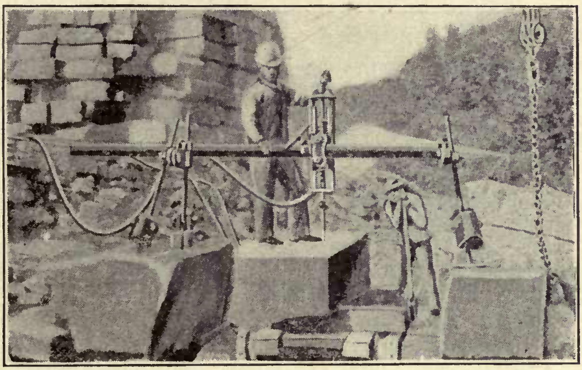


Fig. 21.

or, as shown in Fig. 21, the stone in which the plug holes are being drilled can be moved on a truck. By having wheels with perforated holes (Fig. 21) the drill helper can move the truck, by means of a crow-bar, just far enough for drilling the next hole. It is stated in the Ingersoll-Sergeant catalogue that in a granite a "Baby" drill on a quarry bar will drill a hole 3 or 4 ins. deep in ¾ min., and it can be moved and started in another hole in less than ¾ min., so that 100 ft. of hole are drilled in a day.

The quarry bar is a device that should be used far oftener than it is; for example, wherever vertical drill holes are spaced close together, as in shallow, open cuts, a long quarry bar may be preferable to a tripod, because of the time saved in setting up. In trench work a quarry bar might, in many cases, be used to advantage with the bar spanning the trench.

Broaching.—In quarrying granite the quarry bar is used to some extent for broach channeling, which consists in drilling a row of holes close together like the holes in a postage stamp and then using a "broach," or chisel, to break down the rock between the holes. The wall left between the holes is ¾ to 2 ins. thick, depending upon the hardness of the rock. The "broach" is, of course, not rotated like the ordinary drill bit. In the Ingersoll-Sergeant catalogue the following data are given as to average broach channeling work done per day by one drill on a quarry bar: In granite, 10 to 20 sq. ft.; in marble, 20 to 30; in limestone, 15 to 35; in sandstone, 20 to 40 sq. ft.

Where it is necessary to excavate igneous rocks, like granite or schist, close up to large buildings whose foundations must not be disturbed, broach channeling is often specified.

In this connection it is well to quote from *Engineering Record,* Feb. 7, 1903, a method of blasting close to a tall brick building without channeling. The rock excavation was 60 ft. deep, the rock being stratified in 1 to 4-ft. layers. A trench 10 ft. wide was taken out (10-ft. lifts) parallel with the building. Along the face of the building a row of holes was drilled 18 ft. deep, holes being 6 to 8 ins. center to center. A second parallel row of holes was drilled 2 ft. away from the first row, the holes being 2 ft. apart and loaded lightly with 40 per cent. dynamite. The holes in the row next to the building were not charged, but the blast caused the rock to crack along the line of these uncharged holes.

The Gadder.—The Ingersoll gadder is a machine shown in Fig. 22. It is simply an ordinary rock drill mounted upon a block which can be raised or lowered on an upright post. The post is pivoted at its lower end to a heavy cast-iron bed plate mounted on wheels. The machine will drill holes in a horizontal line near the floor of the quarry, or in a vertical row, or in a line at any desired angle, for the post can be tilted at will. After drilling the holes, plugs and feathers are used to break off blocks as desired. Fig. 22 shows parallel channels made with a channeler, and it shows the plug and feather holes made with the gadder. A drill is said to have a record of 350 ft. of holes in 10 hrs. in marble, only $^1/_3$ min. being required to move from one 2-ft. hole and begin drilling the next.

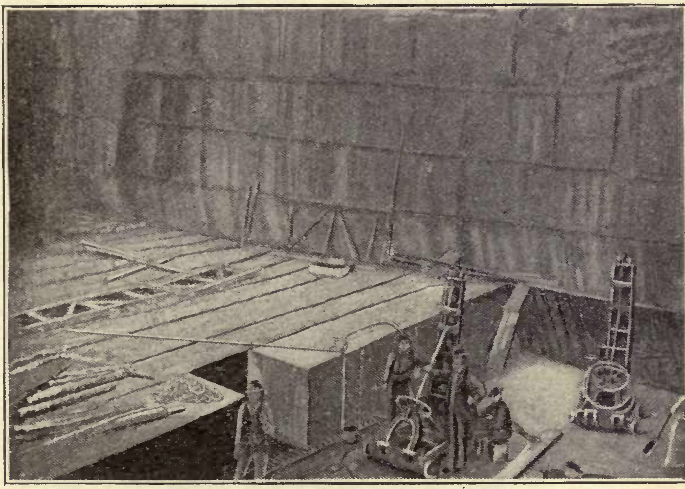


Fig. 22.

Channelers.—In quarrying sandstones and marbles, channeling machines are largely used; and, in the future, channelers will be often used in rock excavation where it is desirable to have smooth sides. On the Chicago Canal excavation track channelers were used. Fig. 23 shows a Sullivan channeler, and Fig. 24 shows a wheel pit extension, 21 ft. wide and 185 ft. deep, made with Sullivan channelers for the Cataract Construction Co. at Niagara Falls. It will be noticed that at each successive lift there is an offset or step of about 6 ins., but that by giving a slight batter to the wall in each lift the trench preserves the same width at the bottom as at the top.

A track channeler is a self-propelling machine that travels back and forth on a 10 to 30-ft. section of track having a gage of 4 ft. 11 ins. The channelers used on the Chicago Canal weighed about 11,000 lbs. each. The stroke was 10 ins., and about 250 blows were struck per min., the channeler moving forward a fraction of an inch at each blow. The gage of the cutting bit was 2⅜ ins. at the start, and decreased in width by ⅛ in. each 2 ft., as in drilling. The extreme depth of a lift was 14 ft. The channels cut were perfectly vertical. Channelers are made that will cut up an

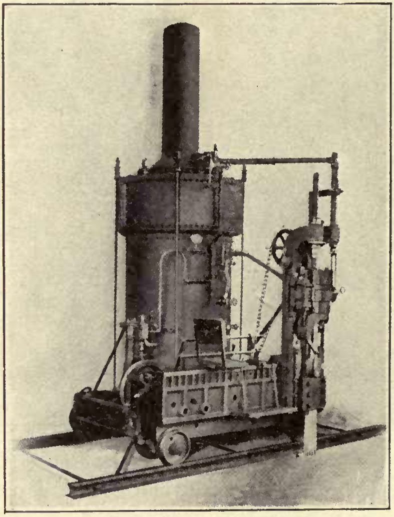


Fig. 23.

angle of 45° for use in quarries where the strata have a sharp dip. Channelers are also made to be mounted on quarry bars, the catalogues of makers showing a variety of types and sizes. Manufacturers state that a 25 to 30 H.-P. boiler will run a channeler having a 6½ to 7-in. cylinder, or that 300 to 350 cu. ft. of free air per min. at 80 lbs. pressure will be consumed. A re-heater is generally mounted in place of the boiler when compressed air is used.

The following data are given in the Ingersoll-Sergeant catalogue as being conservative estimates based upon actual monthly averages. The number of square feet channeled per day: 75 sq. ft. of hard brownstone or sandstone; 75 to 150 sq. ft. of marble; 200 sq. ft. of soft Lake Superior browstone; 250 sq. ft. of soft oolitic limestone. The actual

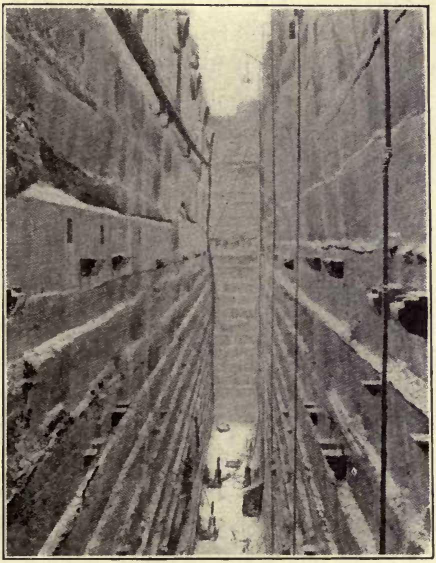


Fig. 24.

averages on the Chicago Drainage Canal work are given in Chapter XIII., where also are given reliable data of cost. It requires two men and about ½ ton of coal per day to run a channeler—in fact, the cost of operating is practically the same as for a steam drill.

It does not pay to use track channelers for quarrying granite, since broach channeling in granite is cheaper.

For quarrying large dimension stones (granite excepted) the channeler has become an economic necessity. Its first cost should not prevent its purchase, once the quarry has been opened sufficiently to prove the marketability of the stone. A channeler will quickly save its cost in the better price received for the stone and in the saving on freight. The last item is one often overlooked, but it may be said, roughly speaking; that fully 20 per cent. of the stone quarried without channeling is lost in the subsequent cutting and dressing after reaching its destination. Since rough dimension stone is paid for by its neat measurement, it is evident that in the end the quarryman must foot the bill for this waste and the freight upon it. The actual cost of channeling when computed in cents per cubic foot of stone is really slight; for the stone is not cut up with the channeler into merchantable blocks, like harvesting ice, but a series of parallel channels are cut across the quarry so as to loosen blocks of stone which may be 50 ft. or more in length. These long blocks are then split with plug and feathers into sizes that the derricks can handle. The smaller blocks are then either sawed up, or still further reduced in size by plug and feathering. If the channels are 9 ft. apart, each square foot of channel releases 9 cu. ft. of stone, so that if the cost of channeling is 9 cts. per sq. ft., the cost per cubic foot is 1 ct.

In dimension stone quarries very large guy derricks are used, so that it is possible to handle blocks of stone weighing 20 tons. The following paragraph gives the cost of one of these huge derricks:

Cost of a Quarry Derrick.—Saunders gives the following cost data in the magazine, *Stone* (New York), 1890, p. 95: A large quarry derrick capable of lifting 20 tons with a single line, having a 24 x 24-in. mast, 75 ft. high, and a 65-ft. boom actually cost as follows:

Timber for mast	$45.00
Timber for boom	28.00
Expense of delivering timber	16.50
Carpenter work on mast and boom at $2.50 a day	25.00

Complete set of derrick irons, sheaves, etc.	$219.00
2,400 ft. best galv. 1-in. iron rope for 8 guys	237.00
Thimbles, clamps, etc.	25.00
500 ft. steel hoisting rope, 1⅝ in.	240.00
Labor on dead men, 4 men, 2 days at $1.40	11.20
Labor raising derrick, 8 men, 2 days, at $1.40	22.40
Labor fixing guys, 8 men, 2 days, at $1.40	22.40
Total	$891.50

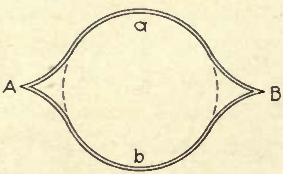


Fig. 25.

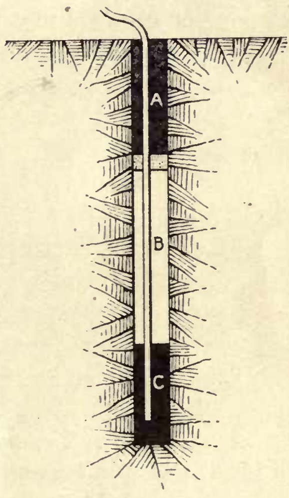


Fig. 26.

Knox System of Blasting.—In *Trans. Am. Soc. C. E.*, 1891, Mr. William L. Saunders describes the Knox system of blasting, named after the inventor. (The patents have recently expired.) The system consists in drilling a number of ordinary round holes in a row and then using a reaming tool to give the hole the shape shown by the heavy lines in Fig. 25. The reaming is done by hand. In medium sandstone the holes may be 10 to 15 ft. apart, but in limestone I find that they are often placed as close together as 4 ft. The holes are charged with black powder, or with Judson powder, as shown in Fig. 26, a wad of hay being put in so as to make an air space between the powder and the tamping. The blast causes the rock to split in a straight line in the direction of the pointed or wedge-shaped sides of the hole. For block-holing, where it is desired to split a block into just four pieces, a single hole is reamed as shown in Fig. 27.

Saunders gives the following data: At Portland, Conn., 15 Knox holes in brown sandstone, charged with 2 lbs. of black powder (No. C) in each hole, loosened a block of rock 110 ft. long, 20 ft. wide and 11 ft. thick, weighing 2,400 tons.

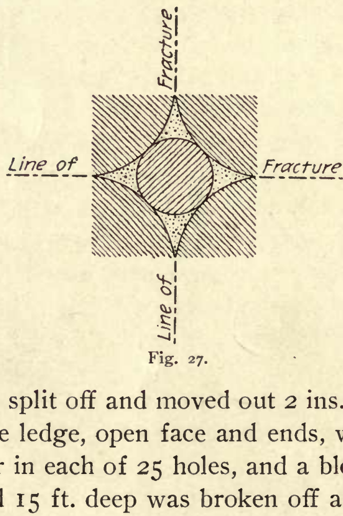


Fig. 27.

This block was split off and moved out 2 ins. *en masse*. Another sandstone ledge, open face and ends, was blasted with 1 lb. of powder in each of 25 holes, and a block 200 ft. long, 28 ft. wide and 15 ft. deep was broken off and moved ½ in. At the mica schist quarries at Conshohoken, Pa., a blast of ½ lb. of powder in a single hole broke a block 27 ft. long, 15 ft. wide and 6 ft. thick across the rift.

Cost of Quarrying for the Buffalo Breakwater.—In *Engineering News* May 16, 1901, Mr. Emile Low (in an article on the Buffalo breakwater) gives data on quarrying by the Knox system. The contractors, Hughes Bros. & Bangs, signed their contract Jan. 27, 1897, at the following prices: Gravel hearting, 13 cts. per cu. yd.; rubble stone, 80 cts. per ton of 2,000 lbs.; capping stone and revetment, $1.25 per ton. No work was done in the winter. Water telescopes were used in placing the revetment. Of revetment, 235 tons were placed daily, the stone weighing 6 2/3 tons each on an average, and up to a maximum of 17 tons. All scows were provided with glass gages and graduated rules for weighing the stone. A gage is made of 3-in. wrought iron standpipe, into which two brass cocks are screwed. Between the cocks, which are 4½ to 7 ft. apart, depending on the draft of the scow, is placed a 1-in. glass tube; and a wooden rule graduated to hundredths of a foot is attached alongside. Lockport limestone weighing 165 lbs. per cu. ft. solid, and Medina sandstone weighing 152 lbs. per cu. ft. solid were used for small rubble. The voids in the broken stone were 50 per cent.

Most of the stone used was a limestone, quarried near Windmill Point, Ontario, and weighed 166 lbs. per cu. ft. The stone was taken out in four ledges; the first, 20 to 36 ins. thick; the second, 6 to 9 ft.; the third, 7 to 10 ft.; the fourth, 5 ft. In opening the quarry a trench 30 ft. deep x 100 ft. wide was made as rapidly as possible, using heavy charges of dynamite. In quarrying the face was worked in four ledges. The top ledge was drilled with holes 18 ft. back from the face and 4 ft. apart, the holes going down to within 6 ins. of the bottom of the ledge. An attempt was made to start and end at a joint, so the ledge could be moved entire for 200 ft. or so. Three sizes of Ingersoll-Sergeant steam drills were used: A (2¼ in.); C (2¾ in.), and F (3½ in.), At a depth of 18 ft. the hole was 1¾ ins. diam., losing ¼-in. every 3 ft. After the holes were drilled they were reamed to an elliptical shape (Knox hole) by a diamond-shaped tool driven either by hand or by steam. Black powder was charged, 3 or 4 handfuls in each hole first; then the exploder, then a little more powder; then a wad of grass was forced down leaving 2 or 3 ins. of air above the charge; then a clay tamping to the top of the hole. Dry sand is sometimes used instead of clay, being more quickly placed, and giving good results. Sand is also easier to clean out in case of misfire. To clean out a misfire hole, a steam hose is attached to small pipe through which steam and water are blown as the pipe descends, thus blowing out the charge.

One block of stone, 180 ft. long, 18 ft. wide and 9 ft. thick, weighing 2,430 tons, was blasted off by one firing, requiring 52 holes, 8 to 9 ft. deep, 18 ft. from the face and 3½ ft. apart, loaded with 75 lbs. of black powder. These 52 holes were loaded and tamped with sand in 1 hr., where it would have taken 2½ to 3 hrs. with clay. The block, (1,080 cu. yds.) was moved 2 ft. by the 75 lbs. of powder, that is, 1 lb. of powder loosened 14 cu. yds. of solid rock.

Another (3,375-ton) block 250 ft. long, 9 ft. thick and 18 ft. wide, was thrown out (2 to 3 ft.) with 150 lbs. of powder in 62 holes

After these large blocks were separated from the ledge, they were split up by drilling holes and using either plug and feathers or light powder charges.

The plant consisted of: 2 A, 9 C and 3 F drills; 8 derricks in the quarry and 1 at the loading dock; 4 50-h.-p. boilers for derricks; 4 skeleton 20-h.-p (8¼ x 12-in.) hoisting engines for 8 quarry derricks; 1 hoist and boiler for hauling cars up incline; 1 boiler for 2 steam pumps for draining quarry; 1 dinky locomotive; 50 cars, 3-ft. gauge; 68 skips, holding 3 to 4 tons, for carrying stone on flat cars; blacksmith shop and 5 forges; machine shop; track, etc.

The force during June, 1903, was as follows:

General:	Rates of wages.	Total per 10-hr. day.
1 superintendent	$167.00 per mo.	$7.00
1 time keeper	60.00 per mo.	2.25
1 general foreman..........	85.00 per mo.	3.25
Stripping gang:		
1 foreman	2.25	2.25

4 laborers	$1.50	$6.00
1 team	3.00	3.00
Quarry:		
1 asst. genl. foreman	3.00	3.00
8 foremen (one per derrick)	2.25	18.00
14 machine drillers	2.00	28.00
14 machine drillers helpers	1.50	21.00
4 hoist engineers (derricks)	1.75	7.00
1 hoist engineer (inclined plane)	1.75	1.75
5 firemen	1.75	8.75
50 laborers	1.50	75.00
1 water boy	1.00	1.00
1 watchman	1.75	1.75
1 team	3.00	3.00
Loading dock:		
1 foreman	2.25	2.25
1 hoist engineer	1.75	1.75
1 fireman	1.75	1.75
6 laborers	1.50	9.00
1 watchman	1.75	1.75
Track repairs:		
1 foreman	2.25	2.25
3 laborers	1.50	4.50
Blacksmith shop:		
1 foreman	3.00	3.00
3 blacksmiths	2.50	7.50
3 helpers	1.75	5.25
Others:		
1 locomotive driver	2.50	2.50
1 machinist	75.00 per mo.	3.00
2 carpenters	1.75	3.50
	Total,	$240.00

TABLE XXI.

Month, 1903.	Stone quarried, tons of 2,000 lbs. Rubble.	Capping.	Total.	Cost of Labor only.	Cost per Ton, cts.
May	16,535.9	......	16,535.9	$5,127.51	31
June	12,771.2	2,541.4	15,312.6	5,154.65	34
July	11,444.4	5,273.8	16,718.2	5,438.91	33
Aug.	9,426.2	5,118.7	14,544.9	5,071.92	35
Sept.	5,937.0	2,931.9	8,868.9	3,283.85	37
Total ...	56,114.7	15,865.8	71,980.5	24,076.84	33

TABLE XXII.

Month, 1903.	Days Worked.	No. of holes drilled. A	C	F	Total.	Lin. ft. drilled. A	C	F	Total.	Explosives, lbs. Powder.	Dynamite.	Coal, tons.
May	24½	513	896	556	1,965	1,385	4,757	4,840	10,982	1,691	302	211
July	24½	674	2,101	674	3,449	1,177	10,771	4,927	16,875	2,683	292	226
Aug.	23.7	620	1,978	658	3,256	853	10,098	4,677	15,628	2,558	117	236

TABLE XXIII.

Month, 1903.	No. Holes per Day per Drill.			Lin. Ft. of Hole per Day per Drill.			Average Depth of Holes.		
	A	C	F	A	C	F	A	C	F
May	12.5	10.0	8.8	33.8	53.4	76.2	2.7	5.3	8.7
July	13.8	9.5	9.5	24.0	49.0	67.0	1.7	5.1	7.3
Aug.	13.3	9.4	9.4	18.4	47.3	65.8	1.3	5.1	7.1

About 1 lb. of black powder was used for every 7 tons of stone quarried, and 1 lb. of dynamite for every 67 tons, and 1 ft. fuse for every 5½ tons of stone.

The cost of powder, dynamite and fuses per ton of stone was: In May, 1. 3 cts.; July, 2.0 cts.; August, 2.1 cts.

The total cost of quarrying stone, loading and placing on scows was as follows:

	Cost per ton, cts.	Cost per cu. yd., cts.
Labor	33	74.3
Coal	4	9.0
Explosives	2	4.5
Miscellaneous	5	11.2
Total	44	99.0

Cost of Quarrying Dimension Sandstone.—In *Engineering News,* Nov. 21, 1901, Mr. R. C. McCalla, Jr., gives the following: In October, 1891, 200 cu. yds. of backing and 600 cu. yds. of dimension stone were quarried for Lock 2, Black Warrior River, Tuskaloosa, Ala. The stone was a fine quality of blue sandstone quarried from the bed of the river at the falls, after diverting the water. The cost of qarrying these 800 cu. yds. was $1,598, or about $1 per cu. yd. for the backing and $2.33 per cu. yd. for the dimension stone. In this month 434 cu. yds. of dimension stone were cut by stone cutters at a cost of $6.83 per cu. yd. The masonry wall is 390½ ft. long, 8 to 14 ft. wide, and 34 ft. high, built in courses of ashlar 18 to 24 ins. thick, and about 50 per cent. cut stone. In October two gangs of masons, using two derricks, laid 1,563 cu. yds. of first-class masonry at a total cost of 92½ cts per cu. yd., including the cost of screening sand, mixing mortar, operating steam hoists, unloading material at the wall and converting them into masonry. The itemized cost of the mason work was:

Foreman, 1 mo			....	$90.00
Masons,	202	days of 8 hrs., at	$2.80 ...	565.60
Laborers,	35⅛	" " "	$1.20 ...	42.15
"	270½	" " "	$1.00 ...	270.50
"	369⅝	" " "	$ .80 ...	295.70
"	146¾	" " "	$ .60 ...	88.05
Boys,	83¼	" " "	$ .40 ...	33.30
Wages paid in board			....	42.00
Fuel for hoists			....	18.49
Total, at 92½ cts. per cu. yd			....	$1,445.79

Quarrying by Water Cushion Blasts.—The following method of quarrying is described in *Engineering Record,* April 7, 1900:

At Cobleskill, N. Y., limestone was quarried for the backing of the East River Bridge piers. Most of the backing is laid in 3-ft. courses; the stone is remarkable for its smoothness, many beds requiring no dressing. The quarry is a solid stratum 28 ft. thick, with vertical fissures at right angles to each other and up to 100 ft. apart. A row of vertical holes 3 or 4 ft. apart is drilled through the stratum from 3 to 10 ft. back of the face, depending on size of blocks required. The holes are filled three-quarters full with water, plugged, and a charge of black powder put in over the plugs and tamped. When fired, a block of solid rock 28 ft. high and perhaps 100 ft. long and 6 ft. thick, was separated and remained standing in its original position. Cross rows of vertical holes were drilled and fired similarly to the first holes, breaking the stone into blocks 10 ft. long and 28 ft. high. These blocks were thrown over and split with plug and feathers into blocks of thickness required for the courses.

Granite Quarrying.—In most granite quarries steam drills, derricks and hoisting engines are the only machines used. In "sheet quarries" after a trench is blasted out to open up a face, if no natural face exists, then the two ends of the face are freed by making channels.

At the Crotch Island quarries, in Maine, two parallel rows of holes are drilled 3 ft. apart, the holes in each row being 8 ins. apart and as deep as the sheet of granite, which varies from 2 to 16 ft. thick. These holes are charged with 60 per cent. dynamite, two sticks at the bottom of each hole, then a plug of wood 8 ins. long on top, then a stick of dynamite 8 ins. long on top of the wood, then another plug of wood, and so on until within 1 ft. of the mouth of the hole, which is tamped. A cap is placed in the last cartridge in each hole, and the holes are fired in pairs. It is not necessary to put a cap in any of the lower sticks as the shock sends them all off practically together. This heavy loading results in tearing the granite into chips which are often hurled a great distance, necessitating blasting at night; but the powdered granite left in the channel, is easily shoveled out, leaving a trench about 4½ ft. wide. Having freed the two ends, a long block of granite, the thickness of the sheet, is loosened by blasting. The granite adjoining the channels when cut into blocks shows no sign of weakness in spite of the tremendous blow received in blasting.

I am indebted to the *Engineering Record* for the following description of the Crotch Island quarry:

A diagram of the method of working is shown in Fig. 28, not made to scale or true dimensions, but merely indicating the operations. 1-2-3-4 and 5 are successive strata of increasing height and from 5 to 12 ft. thick. Suppose that strata 4 is 9 ft. thick and it is desired to quarry from it stones 12 ft. long. On the required line B-H a pair of Lewis* holes about 12 ins. apart and 9 ft. deep are made at

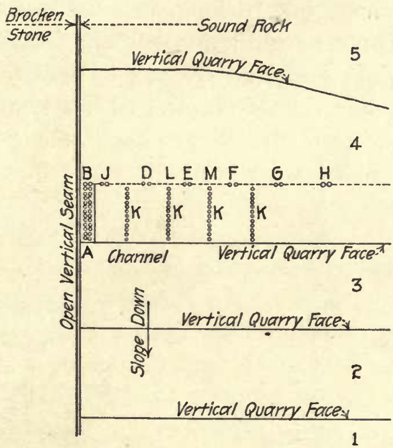


Fig. 28.

D with a compressed air drill and black powder is fired in them; they are swabbed out, recharged and refired, and so on several times until a crack has been opened from J to L. Another similar pair of Lewis holes is made in the line of the crack about 40 ft. away at E and they are similarly fired. Holes are drilled at G-H and so on, and the crack is produced as far as desired, extending everywhere through to the stratum below. Holes 9 ft. deep are drilled 8 to 12 ins.

* Any drill holes placed close together are called Lewis holes.

apart in two rows 3 ft. apart from A to B, adjacent to the open seam which bounds one side of the quarry to form the channel. The pair of holes nearest A are heavily loaded with dynamite and fired as above described, then the next pair are fixed, and so on. Each blast pulverizes the granite between and close to the holes and throws the fragments so far that these blasts are fired at night when only the two men in charge remain on the island. When the whole set of holes has been fired a channel has been formed about 3½ ft. wide which extends through the stratum from A to B and gives a free face. The slab A-B-H-M, 12 ft. wide, 9 ft. thick, and perhaps 300 ft. long, is thus detached from the stratum, but is not moved more than the fraction of an inch from its original position. Blocks of any required width are laid off by lines (K-K-K, etc.) of small holes, which are drilled by hand, and the stone is split along them by the regular plug and feather method.

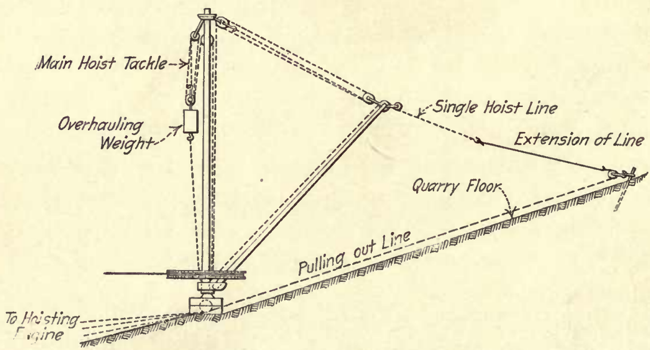


Fig. 29.

The method of removing these blocks is ingenious. Dog holes are made in each and the main line from a derrick, Fig. 29, is attached and a strain is put on by the engine, but does not move it. The crack B-H is perhaps 1/8 to 1/16 in. wide, and in it, opposite the center of the blocks, are poured two cups of thick black oil at points about a foot apart.

Between these points is poured a handful of black powder, which is covered with dirt, and has a fuse attached. The powder does not spread beyond the oil line, and so is confined in a thin sheet, filling the crack from top to bottom. When the fuse is lighted there is a light explosion which does not break the stone, but suffices to kick it out of its place and thus started, is easily pulled by the derrick line, over the smooth, steep slope of the underlying stratum to a point within reach of the derrick boom. There it is split into required sizes by plug and feathers, and the pieces are loaded by the derrick on cars, lowered down inclined track by cables to the two docks where other derricks load them on schooners for shipment.

Quarrying Massive Granite.—When granite does not occur in beds or sheets of moderate thickness, the method just described requires some modification. The method practiced at Mt. Airy, in North Carolina, according to Merrill, is as follows: No quarry face is used, but a hole is drilled in the massive granite perpendicular to the surface, and to a depth of 6 to 12 ft., according to the thickness of the stone desired. This hole is loaded with a light charge of powder and fired, then it is loaded with another light charge and fired, and so on until cracks appear in the granite at the surface at a distance of 150 to 200 ft. from the hole, caused by the lifting bodily of a lense-shaped mass of the granite by the force of the powder. The blasting is repeated until the lense-shaped mass is almost free all around, when it is left for a day or two so that the stresses produced by the changes of temperature from day to night break the mass of granite entirely free. Then this lenticular mass is split with wedges into blocks that can be removed. This is a very ingenious method and one well worthy of introduction wherever granite occurs in massive form, but the most common method is as follows:

The granite is blasted out in large, irregular chunks, using as small charges of powder as will effect the loosening of the rock, the method being in fact similar to ordinary open cut excavation described in Chapter XII. The largest and most regular blocks are selected for splitting up with plug and feathers, and the other stones are used as far as possible for rubble or concrete. In upper New York, on the Spier Falls Dam construction, and in Massachusetts, on the Wachusett Dam work, I have seen this method used on a large scale. On a smaller scale I have used it myself, but in all cases the cost of the dimension stone so secured has been excessive. The Adirondacks "granite" is an exceedingly tough stone, but in spite of the greatest care I have had cut stone break in two while handling them, because of the shattering effect of the dynamite; and black powder did not prove much more satisfactory. On both the large dams just mentioned it was found cheaper to import cut stone long distances than to use the local granite for certain parts of the cut stone work. If granite boulders occur, they can be split up with plug and feathers yielding splendid blocks. In fact the early qarrying in a granite region is apt to be boulder quarrying; and, in consequence, quarries of massive granite that is blasted out in rough chunks and split with plugs and feathers, are often called boulder quarries.

An effective way of making "boulders" is by large chamber blasting, where the amount of stone required warrants quarrying on such a large scale.

Cost of Quarrying Granite.—Cost data relating to the quarrying of granite dimension stone are extremely hard to secure. I have been able to find only one writer, Mr. J. J. R. Croes, who has published anything on the subject. Mr. Croes' records, together with mine, will at least form a basis for approximate estimates of cost of granite quarrying. My data apply to quarrying three-dimension stone in a sheet quarry on the coast of Maine. The total number of men engaged was, on the average: 6 enginemen, 6 steam drillers, 6 drill helpers, 3 blacksmiths, 3 helpers, 5 tool and water boys, 38 quarrymen, 47 laborers, 2 foremen and 1 superintendent.

This force quarried and loaded on boats about 1,400 cu. yds. of rough granite blocks. The stone was loaded by derricks onto cars, from which it was unloaded into boats ready for shipment. The following cost includes everything except interest and depreciation of plant, and development expenses:

	Cost. per cu. yd.
Enginemen, at \$2 a day (of 9 hrs.)	\$0.20
Steam drillers, at \$2.00	0.20
Drill helpers, at \$1.50	0.15
Blacksmiths, at \$2.75	0.14
" helpers, at \$1.75	0.09
Tool and water boys, at \$1	0.16
Quarrymen, at \$1.75	1.09
Laborers, at \$1.50	1.15
Foremen, at \$3	.15
Superintendent, at \$8	.20
Coal, at \$5 ton	.45
Explosives	.25
Other supplies	.30
Total	\$4.53

On the best month's work, when a larger force was being operated, the cost of all labor, superintendence and supplies, was reduced to a little below \$4 per cu. yd.; but the above, \$4.50 per cu. yd., may be taken as a fair average of several months' work. To this should be added the charges for plant rental, quarry rental (if any), stripping (if any), and freight charges to destination. The freight rate by boat from Maine to New York is about \$1 a ton, but as rough granite blocks are always measured on their least dimensions, the freight charges when \$1 per ton amount to about \$2.70 per cu. yd., of three-dimension stone in the rough. The explosives used were black powder, costing \$2.25 a keg (25 lbs.), and dynamite for channeling, costing 15 cts. a lb. The sheet from which this granite was quarried averaged about 6½ ft. thick, and was nearly flat. The stone was loosened in long blocks by Knox blasting with black powder, and was split up into sizes by plug and feathering; both hand drills and pneumatic plug drills being used for this purpose. The stone, as before stated, was three-dimension stone. To quarry random stone (not rubble) in this quarry cost about $3.50 per cu. yd.

Cost of Quarrying Gneiss.—Brief, but reliable data on the quarrying of stone for the Boyd's Corner Dam, near New York City, are given by Mr. J. J. R. Croes, in *Trans. Am. Soc., C. E., 1875.* The stone is a gneiss that is found in and about New York City, and containing so much mica that it is more properly called mica-schist. The face stone for the dam average 1.8 ft. rise, 3.6 ft. long and 2.7 ft. deep, and were cut to lay ¾-in. joints. In quarrying the dimension stone, plug and feathers were used to split the stone to size ready for cutting. The cost of quarrying and plug and feathering 4,000 cu. yds. of dimension stone ready for cutting was as follows:

	Days (10-hr.) per cu. yd.	Cost per cu. yd.
Foreman, at $3	0.114	$0.34
Drillers, at $2	0.917	1.84
Laborers, at $1.50	0.429	0.65
Blacksmiths, at $2.50	0.102	0.25
Tool boys, at $0.50	0.108	0.05
Labor loading teams, at $1.50	0.284	0.42
Total (not including explosives and teaming)		$3.55

The work was done by contract in 1867-8. The rates of wages were not given by Mr. Croes, but Mr. John B. McDonald has been kind enough to give me most of the rates of wages as nearly as he can remember. The length of haul from quarry to stone yard was about a mile, and Mr. McDonald states that oxen were used. The cost of "teams" is given by Mr. Croes as 0.62 team days per cu. yd., which indicates that a good deal of stone boat work was done, or else that there is an error in this item.

The cost of quarrying 3,400 cu. yds. of rubble stone for this same dam was as follows:

	Days per cu. yd.	Cost per cu. yd.
Foremen, at $3	0.041	$0.12
Drillers, at $2	0.339	0.68
Laborers, at $1.50	0.140	0.21
Blacksmiths, at $2.50	0.036	0.09
Tool boy, at $0.50	0.035	0.02
Labor, loading teams, at $1.50	0.077	0.12
Teams, at $4	0.141	0.56
Total labor		$1.80

It is presumable that both the dimension stone and the rubble stone were measured in the dam.

Cost of Quarrying Sandstone.—In quarrying thin bedded sandstone for dry slope walls and rubble, I have found that one quarryman will average about 2 cu. yds., per 10-hr. day. In doing this work no powder is used where the beds lie free, but if they are cemented together it is necessary to shake up the ledge with light charges of black powder. Wedges, crow-bars and hammers are the only tools needed for quarrying thin bedded stone where the beds can be separated by driving wedges in between them. The stone quarried thus is not very regular, except on the bed joints; and, when it is dressed up by the mason, there is a considerable shrinkage in volume between the measurement of the stone corded on a stone rack and the stone measured in the wall. The mason uses the spalls to fill in the vertical joints, so that there is little or no real loss of stone. In quarrying several thousand cu. yds. of stone for dry slope wall masonry, I found that 2 cu. yds. of stone, measured corded on the wagon, made 1.55 cu. yds. of slope wall. Each quarryman averaged 2 cu. yds. per day of stone as measured in the wall, or 2.6 cu. yds., measured corded in wagons. These quarrymen received $1.75 a day, and as practically no powder was used, the cost was 88 cts. per cu. yd. for quarrying stone measured in the wall, and this included loading onto wagons, but not hauling.

Cost of Quarrying Limestone.—Mr. James W. Beardsley is my authority for the following data on the cost of quarrying limestone for retaining walls on the Chicago Canal. The contractors selected parts of the canal where the limestone occurred in strata that were uniform, so that the beds of the stone quarried required no dressing. The stone was laid in courses averaging about 15 ins. thick, the better stone being selected for the face of the wall. Guy derricks having a capacity of 6 to 10 tons, boom 40 to 60 ft. long, operated by a hoisting engine, were used for loading the stone. Black powder was used to shake up the ledges, and the stone was then barred and wedged out. The cost per cu. yd. is the average of 93,500 cu. yds., measured in retaining walls. The mortar was only 13¼ per cent. of the wall, indicating an unusually even bedded stone that squared up well. The cost does not include general superintendence, installation of plant, plant rental, powder, material for repairs, and cost arising from delays.

	Typical Force.	Wages per ten hrs.	Per cent. of cost.	Cost cu. yds., cts.
General foreman	0.01	$4.75	00.2	00.2
Foreman	1.00	3.50	10.6	7.8
Derrickmen	2.11	1.50	10.1	7.5
Quarrymen	8.42	1.65	42.2	31.2
Enginemen	1.10	2.25	7.0	5.2
Firemen	0.04	1.75	0.2	0.2
Laborers	2.28	1.50	10.9	8.0
Water boys	0.33	0.75	0.9	0.7

Blacksmiths	0.27	2 to 3	1.7	1.3
" helpers	0.18	1.75	0.9	0.7
Carpenters	0.02	2.25	0.1	0.0
Drill runners	0.36	2.00	3.1	2.3
" helpers	0.07	1.50	0.4	0.2
Watchmen	0.04	1.50	0.1	0.1
Teams and carts	0.29	3.50 and 2.50	3.8	2.8
Derricks	1.12	1.35	5.4	4.0
Drills	0.36	1.15	2.1	1.5
Total	16.52 (men)		99.7	73.7

This cost of 73.7 cts. per cu. yd., it should be borne in mind, is the cost of quarrying rubble stone occurring in regular beds. The cost of quarrying Manhattan gneiss and sledging into sizes fit for rubble is given on page 222.

CHAPTER XII.

OPEN CUT EXCAVATION.

General Considerations.—In this chapter will be discussed all open cut rock work except trenching, and building stone quarrying.

The removal of the earth "over burden," as the English call it, or the "stripping" is it is termed in America, is discussed in my book on earthwork, so that no space will be given here to that factor of cost. In selecting a quarry site, of course, the character and depth of stripping should always receive careful consideration, bore holes and test pits being sunk to ledge rock. Another feature that should never be overlooked is the drainage. A pit dug below the level of the lowest natural drainage channel will often make excavation exceedingly expensive where much water flows or seeps into it, and in winter it may drift full of snow, making work impracticable. I have opened several small quarries in the bed of streams that run nearly dry in summer, for in such places the stripping is likely to be slight. Quarries are preferably located in the side of a hill where gravity drainage will be secured. Moreover, in such a location there is generally no need of snatch teams or hoisting engines to haul the wagons or cars out of the pit; whereas, in a pit below the level of the surrounding country there is a constant outlay of money for raising the excavated rock.

Excavation in Benches.—In deep, open cuts or pits, the rock is usually excavated in two or more benches or lifts. On the Chicago Canal, for example, the rock cut was about 36 ft. deep, and it was taken out in three 12-ft. lifts. There are two factors that determine the economic height of a lift: (1) The depth to which the drill will bore economically, and (2) the size into which the rock breaks upon blasting.

With the ordinary 3⅛-inch drill, about 16 to 20 ft. is the limiting depth of economic drilling, but with a 3½-in. drill it is often economic to drill 24 ft. If a well drilling machine is used, it is possible to go down 100 ft. or more, and in fact, I have seen a shale bench 60 ft. high taken out where a well driller was in use. The height of the bench, however, is not dependent solely upon the economic depth of drilling. The higher the bench the farther back from the face may the row of drill holes be located; but the farther back that the drill holes are placed, the larger will be the chunks of rock thrown down upon blasting, unless the rock is exceedingly friable like shale. If, in a hard limestone, for example, the bench is 25 ft. high, and the drill holes are placed in a row only 9 ft. back of the face, the blast may blow out the bottom of the bench, and leave the top overhanging; and even if the top were to fall it would come down in very large chunks, although the bottom might be broken up to the desired size. This objection to a high bench with drill holes close to the face may be overcome by separating the charge in each hole into two or more parts, with tamping between; and, as a matter of fact, I am surprised that this is not done oftener.

We have seen in Chapter V. that it pays to drill as deep a hole as the capacity of the drill will permit in order to reduce the time lost in moving from hole to hole. It should be added that the depth of the hole should ordinarily be some multiple of 2 ft. (if the feed screw is 2 ft. long), for once a new bit is in place it should be made to drill as far as the feed will permit. This rule should be ignored where other reasons prevail; thus, in stratified rock it is often well to stop the drill hole just short of a seam of stratification.

In order to increase the height of the benches, a common expedient is to drill one or two rows of horizontal holes in the face of the bench, as well as the row of vertical holes, as shown in Fig. 32. This can ordinarily be done only where the bench is long enough to permit drillers to work on the floor at one place while the loaders are working at another place. This is a good expedient to employ where one portable drilling plant is used to work two or more quarries, for in this way a high bench can be blown down at one blast, instead of taking it down in two shallow benches, and thus time is saved in moving the drilling plant. It is also an expedient often used in side hill cutting where a steam shovel is used for loading.

Spacing Holes.—A common rule is to place the row of vertical drill holes a distance back from the face equal to the depth of the drill hole, and to place the drill holes a distance apart in the row equal to their depth. Another rule is to place the row of holes back from the face a distance equal to three-fourths their depth, and the same distance apart in the row. In stratified rock of medium hardness these rules may be followed in making the first experiments, but they will lead to serious error if applied to dense granitic rocks. In the limestone on the Chicago Canal, not much of which was loaded with steam shovels, the holes were usually 12 ft. deep and placed in rows about 8 ft. back of the face and 8 ft. apart. These holes were charged with 40 per cent. dynamite. In a railway cut through sandstone the holes were 20 ft. deep, 18 ft. back from the face and 14 ft. apart in the row. These holes were "sprung" three times, and each hole charged with 200 lbs. of black powder. In granite quarried for rubble for dam work, I have had to place the holes 4½ to 5 ft. back of the face and the same distance apart, the holes being 12 ft. deep, about 2 lbs. of 60 per cent. dynamite being charged in each hole. On railway work in the Rocky Mountains about the same spacing was found necessary in granitic rock that was to be broken up into chunks that a steam shovel could handle. In pit

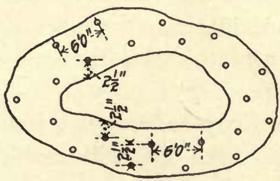


Fig. 30.

mining at the Treadwell Mine, Alaska, the holes are drilled 12 ft. deep, in rows 2½ ft. apart, the holes being 6 ft. apart in each row and staggered, as shown in Fig. 30. This requires drilling 1.7 ft. of hole per cu. yd. I am indebted to Mr. Robt. A. Kinzie for this information. The ore is a tough syenite, and the holes are spaced closer together than would be necessary if the crushers were large enough to receive bigger chunks. In crushing ore or rock on a large scale the mining man and the contractor should bear in mind that it is poor economy to install small crushers, especially where the rock is so tough that it breaks out in large chunks; for a small crusher means not only money lost due to drilling holes close together, but it usually means labor and powder expended in sledging and blockholing the rock before it will enter the crusher.

It is obviously impossible to lay down any hard and fast rule for the spacing of drill holes. In stratified rock that is friable, and in traps that are full of natural joints and seams, it is often possible to space the holes a distance apart somewhat greater than their depth, and still break the rock to comparatively small sizes upon blasting. In tough granite, gneiss, syenite and in trap where joints are few and far between the holes may have to be spaced 3 to 8 ft. apart, regardless of their depth, for with wider spacing the blocks of stone thrown down by blasting will be too large to handle with ordinary appliances. The mica-schist, or gneiss, of Manhattan Island is a good example of rock that requires close spacing of holes regardless of depth. I have seen holes in it 20 ft. deep and only 4 ft. apart.

The effect of spacing of holes upon the cost of excavation is best shown by tabulation of the feet of hole drilled per cubic yard excavated, as shown in Table XXIV.:

TABLE XXIV.

Distance apart of holes, ft.	1	1.5	2	2.5	3	3.5	4	4.5	5
Cu. yds. per ft. of hole...	0.04	0.08	0.15	0.23	0.33	0.45	0.59	0.75	0.93
Ft. of hole per cu. yd.....	27.0	12.0	6.8	4.3	3.0	2.2	1.7	1.33	1.08
Distance apart of holes, ft.	6	7	8	9	10	12	14	16	18
Cu. yds. per ft. of hole...	1.33	1.80	2.37	3.00	3.70	5.32	7.25	9.52	12.05
Ft. of hole per cu. yd.....	0.75	0.56	0.42	0.33	0.27	0.19	0.14	0.11	0.08

Since in shallow excavations the holes can seldom be much farther apart than 1 to 1½ times their depth, we see that the cost of drilling per cubic yard increases very rapidly the shallower the excavation. Thus an excavation 2 ft. deep, with holes 2 ft. apart, requires 4.3 ft. of drill hole per cubic yard, as against 0.42 ft. of hole per cu. yd. in a deeper excavation where drill holes are 8 ft. apart. Failure to consider this fact ruined one contractor on the Erie Canal deepening, where rock excavation was only 2 ft. deep. Furthermore, as we have seen in Chapter V., the cost of drilling a foot of hole is much increased where frequent shifting of the drill tripod is necessary.

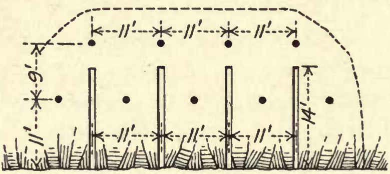


Fig. 31, Plan.

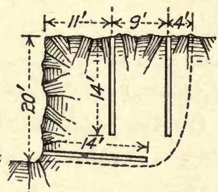


Fig. 32, Profile.

Sometimes granites and traps, even though tough, will break up under the blast for several feet back of the last row of drill holes. Fig. 31, for example, shows a trap rock quarry in which the holes averaged 10 ft. apart and 14 ft. deep, but the rock was full of joints and broke readily, as is indicated in Fig. 32, which shows that the rock broke 3 to 4 ft. beyond the drill holes when the dynamite exploded. Sedimentary rocks often break for much greater distances back of the last row of holes; and this is especially so when the holes have been sprung and charged heavily with black powder. The higher grades of dynamite are more shattering in the immediate vicinity of the drill holes, but are not so apt to break the rock up a short distance away. Joveite of the same strength as dynamite appears to be somewhat slower, in consequence of which it shatters the rock for a greater distance from the hole.

By observing carefully the appearance of rocks in different localities it is possible in a short time to become tolerably proficient in the art of estimating the probable distance apart that holes must be drilled for the best effect with given charges of given kind of explosive; and with this end in view a young man should avail himself of every opportunity of studying prevailing practice in spacing drill holes in different localities.

Cost of Quarrying Trap for Macadam.—The following is an average of the cost of quarrying in several different trap quarries, taken from my own records. The trap was hard to drill, being seamy, but in consequence of its seaminess it broke up well when 75 per cent. dynamite was used. Holes averaged 14 ft. in depth, and three of these holes per day of 10 hrs. were drilled with a 3¼-in. drill using steam from a portable boiler. The cost of operating the drill was:

1 driller	$3.00
1 helper	2.00
1 fireman	2.00
*20 bits sharpened at 5 cts	1.00
1/3 ton soft coal	1.25
Hauling water for boiler	.75
Oil and waste	.25
Drill interest and maintenance	.75
Boiler do	1.00
Total	$12.00

The wages paid were high, and the number of feet drilled low, so that the cost of drilling was nearly 30 cts. per ft. of hole. It required about 0.35 ft. of hole per cu. yd. of rock (solid measure), so that the cost was 10 cts. per cu. yd. for drilling. Common laborers, at $1.50 a day, sledged the rock to sizes that would enter a 9 x 16-in. crusher, and threw the stone back away from the quarry face ready to be loaded and hauled to the crusher. Much of the rock was already

* At a custom blacksmith shop near the work.

broken small enough by the blast, so that a man averaged 7½ cu. yds. (solid) a day sledged and thrown back. The items of cost per cu. yd. (solid) were as follows:

	Cts. per cu. yd. (solid).
Stripping	5
Drilling	10
Sledging	20
Dynamite (75 p. c.), $^1/_5$ lb. at 25 cts. in the hole	5
Total	40

This is the cost per cu. yd. solid (not including loading and hauling away), but 1 cu. yd. solid makes about 1.7 cu. yds. when broken; hence the cost of quarrying was about 24 cts. per cu. yd. measured after breaking. In loading this broken stone into carts one man would load about 15 cu. yds .(measured loose) per day. A man will load into wheelbarrows and wheel a distance of 100 ft., dump and return, at the rate of 10 cu. yds. (measured loose) per day. Further data on the cost of transportation will be found in Chapter X. It will be noted that these data apply to quarrying on a small scale, with a portable plant, in tough rock, but that the rock was seamy and the face fairly high (14 ft.), as shown in Fig. 32. Two men pumping out drill holes and carrying dynamite to two men charging consumed an hour in charging six holes with 50 lbs. of dynamite, or at the rate of 12½ lbs. per man per hour, or about 1½ cts. per lb. of dynamite for charging, tamping and firing.

Cost of Quarrying and Crushing Limestone.—I have had occasion to open limestone quarries where a face only 5 or 6 ft. high could be worked without doing a great deal of stripping. The following was the average labor cost of quarrying and crushing 60 cu. yds. loose measure, or 35 cu. yds. of solid rock, per 10-hr. day, the average being that of 4,000 cu. yds.:

Quarry.		Crusher (9 x 16 ins.).	
1 driller	$2.50	1 engineer	$2.50
1 helper	1.50	2 men feeding	3.50
1 man stripping	1.50	6 men wheeling	9.00
4 men quarrying	6.00	1 bin man	1.50
1 blacksmith	2.50	1 general foreman	3.00
Total	$14.00	Total	$19.50

The "four men quarrying" barred out and sledged the blasted stone to sizes that would enter the crusher; the "six men wheeling" wheeled it in barrows about 150 ft. to the crusher, and delivered in on a platform. The dynamite used was 40 per cent., at 12 cts. per lb., and 0.4 lb. was used per cu. yd. of crushed rock, or 0.7 lb. per cu. yd. of solid rock. One electric exploder, costing 3 cts., was used per pound of dynamite. A long ton of coal, at $2.50, and a gallon of oil, at 25 cts., were used per shift for both crusher and drill. Holes were drilled about 5 to 6 ft. apart, the face being 5 to 6 ft. high, and the drill averaged 60 ft. per shift. Summarizing we have:

Wages of quarry crew	$14.00
Wages of crusher crew	19.50
24 lbs. of dynamite with exploders, at 15 cts.	3.60
1 ton of coal	2.50
1 gallon of oil	.25
Total	$39.85

This is equivalent to 65 cts. per cu. yd. of crushed stone measured in the bins, which is a high cost, due to the low quarry face, and the small plant operated. Current repairs to the drill cost 1 ct. per cu. yd. and the crusher another 1 ct.; steam hose, drill steel and sundry supplies cost another 1½ cts. per cu. yd. of loose rock. Quarry rent was 5 cts. per cu. yd.

Cost of Pit Mining, Brewster, N. Y.—In the magazine, *Stone* (New York), 1892, page 414, Saunders gives the following data of cost of ore bank blasting in an open cut at the Croton Iron Mines. The work was done under Mr. Charles Vivian, contractor, between July 13, 1891, and Jan. 5, 1892:

Total cu. yds. rock and ore	9,295
Total number of drill holes	238
Total feet drilled	2,988
Average depth of hole, ft.	12.50
Ft. of hole per cu. yd.	0.32
Lbs. of 52 p. c. dynamite per cu. yd.	0.44
	Cost per cu. yd.
Labor of all kinds	$0.613
Explosives	.081
Steam for drills	.028
Repairs and supplies	.015
Total	$0.737

This cost includes blockholing and sledging the ore to 7-in. cubes, and the rock to 10-in. cubes. A baby drill was used for blockholing. Mr. Vivian writes me that his original records have been lost, so that the rates of wages cannot be ascertained, but I think it is probably safe to assume that machine drillers received about $2.75, and common laborers $1.25 to $1.50 per 10-hr. day.

Cost of Excavating Gneiss.—I am indebted to Mr. John J. Hopper, civil engineer and contractor, for the following data, part of which originally appeared in the magazine, *Stone*. The work involved the excavation of 29,295 cu. yds. of gneiss (or mica schist) at One Hundred and Twenty-seventh street, New York City. The drilling of the main holes was done with four 3½-in. Ingersoll steam drills, and two "baby drills" were used for drilling block holes. The average height of the lifts was 12 to 15 ft., and the cut ranged from 2 to 63 ft. deep. Hand drillers and sledgers received $2 per 10-hr. day; laborers handling stone and loading wagons received $1.50; one of the machine drillers received $3, and the rest of the drillers received $2.75 a day.

The baby drills were used only on the largest pieces thrown down by the blast; the ordinary sized stone from the blast was broken up by hand-drilled holes and by sledges to sizes suitable for building rubble foundation walls. A good deal of the stone was piled up during the winter until it could be sold. The drilling part of the plant cost $1,800; the boilers, derricks, hoists, etc., cost $1,080; 40 per cent. dynamite, costing 20 cts. per lb., was used. There were 18,433 lin. ft. of main holes drilled (not including block holes) in excavating 29,295 cu. yds. of solid rock. The total cost of the work, including the plant, cartage, sledging, etc., was $52,-635. The itemized cost was as follows:

	Cts. per cu. yd.
Foremen and timekeepers	8.0
Engineers and drillers	10.9
Sledgers	38.3
Derrickmen and helpers	9.6
Labor, loading, etc.	24.7
Hand drillers	11.7
Blacksmith and helper	5.3
Hauling away in wagons	40.5
Explosives	9.8
Coal, coke, oil, etc.	6.0
Repairs to drills	1.0
Repairs to boilers, derricks, etc.	1.2
Total per cu. yd	$1.67

Mr. Hopper informs me that in sound rock where 20-ft. holes could be drilled, a drill would average 70 ft. in 10 hrs.; but in shallow drilling the drills would frequently not average over 25 ft. each.

Cost of Excavating Sandstone and Shale.—In excavating shales and sandstones of the coal measures of Pennsylvania, Ohio, Virginia, etc., I find that holes are usually 20 to 24 ft. deep, and spaced 12 to 18 ft. apart. On an average we may say that for every cubic yard of solid rock there is $^1/_{10}$ lin. ft.

of drill hole, when cuts are very wide, covering large areas of ground; but in thorough cuts for railroads it is not safe to count upon much less than $^2/_{10}$ ft. of drill hole per cu. yd. The holes are almost invariably sprung with 40 per cent. dynamite and then charged with black powder. As low as $^1/_{50}$ lb. of dynamite per cu. yd. may be used for springing holes in shale, and as high as ½ lb. per cu. yd. in sandstone that is to be very heavily loaded. (See page 149). I should put the average at $^1/_{20}$ lb. of dynamite per cu. yd. of shale, and $^1/_{10}$ lb. per cu. yd. of sandstone. A very common charge is 8 kegs (200 lbs.) of black powder per hole, or about 1 lb. per cu. yd. in side cuts, and 1½ to 2 lbs. per cu. yd. in thorough cuts, although as high as 3 lbs. per cu. yd. have been used in thorough cuts in sandstone where special effort was made to break up the rock to small sizes for steam shovel work. The drilling of the deep holes costs not far from 40 cts. per lin. ft. where drilling is done by hand with wages at 15 cts. an hour, and it may be as low as 12 cts. a lin. ft. if well drillers are used. Soda powder costs about 5 cts. per lb., and 40 per cent. dynamite 12 cts. per lb. We have, therefore, the following:

	Cts. per cu. yd.
Drilling $^1/_{10}$ ft. to $^2/_{10}$ ft. at 40 cts.	4.0 to 8.0
Dynamite $^1/_{20}$ lb. to $^1/_{10}$ lb.	0.6 to 1.2
Powder, 1 lb. to 2 lbs.	5.0 to 10.0
Total for loosening the rock	9.6 to 19.2

The rock is commonly loaded with steam shovels, and it is not safe to count upon more than 500 cu. yds. of shale, or 250 cu. yds. of sandstone per shovel per 10-hr. shift. The daily (10-hr.) cost of operating a shovel will average about as follows:

1 foreman	$4.00
1 engineman	3.50
1 craneman	3.00
1 fireman	1.75
6 pit men	9.00

1 locomotive driver	$3.00
1 trainman	2.00
4 dump and trackmen	6.00
1 pumpman	2.50
1 $^1/_3$ tons coal, $4, and oil, 25c.	4.25
Total	$39.00

For plant rental and repairs from $5 to $15 a day must be added, making a total of, say, $50 a day, or a cost of 10 cts. per cu. yd. for soft shale to 20 cts. for sandstone for loading and transporting a short distance, such as 1,000 ft. Where grades are steep and distances long, two or more dinkey locomotives will be required. The resistances of rolling friction and grades are discussed in my book on earthwork, and need not be repeated here. We see that the cost of loosening, loading and transporting shale from side cuts may be as low as about 20 cts. per cu. yd., and sandstone from thorough cuts may be as high as 40 cts. per cu. yd., wages and prices being as above given, and hand drills used for loosening. The cost of moving the plant to and from the work is not included in the above costs.

Other data bearing on steam shovel work will be found in Chapters X. and XIII. It is necessary to study all the conditions of each case and to apply the data given in various parts of this book. I have merely given the above as an example of shale and sandstone excavation on railway work.

Cost of Railroad Excavation in Tennessee.—For the following data I am indebted to Mr. Daniel J. Hauer, C. E., an engineer and contractor experienced in railroad construction:

"In railroad work it is difficult to keep separate records of the cost of excavating earth, loose rock and solid rock. Specifications differ as to classification of excavated materials and engineers differ in their interpretation of specifications, all of which must be borne in mind when studying data of railroad excavation. The costs that follow apply to railroad work done in the Cumberland Mountains of Tennessee.

Under "loose rock" were included shale, slate, coal, soft friable sandstone, cemented gravel, stratified stone in layers less than 6 ins. thick and boulders not less than 2 cu. ft. nor more than 1 cu. yd. in size. Solid rock included all "rock in place which rings under the hammer," except rock in layers less than 6 ins. thick. The clause relating to 6-in. layers was not enforced, but such rock and slate were classified as solid rock. The excavation was all done by hand, there being no power drills or steam shovels used. The rock was hauled to the embankments in barrows, carts and cars. Wages ranged from $1.25 to $1.50 per day; the day being 10 hrs. long in winter and 11 hrs. in summer; the average wage being about 13½ cts. per hr. Laborers were scarce and inclined to be independent. Foremen received $3 a day, or an average of 28.6 cts. per hr. Black powder cost $1.22 per keg (25 lbs.); 40 per cent. dynamite, 11¾ cts. per lb.; Judson powder, 7¾ cts per lb.; double tape fuse, 42 cts. per coil of 100 ft.; quintuple caps, 75 cts. per 100, and electrical exploders, 4 to 7 cts. each, according to lengths. The men were worked in gangs of about 10 men under one foreman, the dumpmen and cart drivers being included in this number. The drivers' wages are included in the cost of "teams," there being one driver to two one-horse dump carts on short hauls, and one driver to three carts on long hauls. Two carts and a driver were paid $3 a day. When dump cars were used, two cars, one mule and a driver were counted at $2 a day. By dividing the number of loads (tallied by the dumpman) into the yardage given by the engineers in the monthly estimates, the following results were obtained:

Dump cart, without tail gate....	Earth	0.6 cu. yd.
" " " " "	Rock	0.35 " "
Dump car, " " "	Earth	1.0 " "
" " " " "	Rock	0.6 " "

Dump car, with tail gate........	Earth 1.25	cu. yd.
" " " " "	Rock 0.7	" "

Tail gates were not used in carts until the haul became 1,200 ft. or more; and tail gates were not used in cars until the haul became 1,500 ft. The dump car body held 1½ cu. yds. water measure. It is safe to count upon loads 10 to 15 per cent. less than those above given; thus, in a train of two or three cars, without tail gates, count on ½ cu. yd. of solid rock per carload. In short hauls, the driver takes one cart or car to the dump while the other is being loaded.

All items of cost are given, except the salaries of superintendent, time keepers, blacksmith and night watchmen; but the cost of these items was 6 per cent. of the total cost, being distributed as follows:

Superintendent	$975.50
Blacksmith	586.80
Time keeper	584.85
Night watchman	457.50
Total	$2,604.65

Each cut was opened up with wheelbarrows, and when the extreme haul for these became 50 ft., either dump carts or cars were substituted. The cars were operated on wooden rails, made of 2 x 4-in. scantlings, either of oak or beech. The entire cost of these tracks is included with labor, except the cost of the two by fours, which were used several times, these being only 10,000 ft. B. M., bought at a cost of $10 per M. These tracks worked well except in dry weather, when it was necessary to have a man pour water on the rails, or else the cars were frequently derailed. On sharp curves and at switches, guard rails were used.

The cost of trimming and dressing up the work is included in each case but it may be of interest to consider some features of it separately. All cuts were excavated a foot below the cross-section stakes, and then this foot was filled back, leaving a ditch on either side of the cut wherever the plans called for one. If this back filling was made from the cut no payment was made for the work; but if it was done with material from a borrow pit, it was paid for as earth. Blue prints were furnished for this work and stakes were driven to grade, and the work was supposed to be done within 0.05 ft. This method saved the railroad company a little money in ballasting, but it added materially to the cost of dressing up for the contractor. He was allowed 6.6 cu. yds. of earth on a 10° curve (the maximum curve used) per 100 ft. of roadbed, for putting on this elevation. On lighter curves a less amount was allowed. In nearly all cases the cuts were taken out a foot below grade, and the embankments were high; yet to dress up 6,300 lineal ft. of roadbed cost $1,-226.14, making a cost of 0.97 cts. per sq. yd., or 1½ cts. per cu. yd. of material moved within the 6,300 ft. This cost is excessive, especially when slopes of cuts have been trimmed as work progresses, so that only the roadbed has to be dressed. Such work should have been trimmed and dressed for less than ½ ct. per sq. yd.

Table XXV. gives the yardage and the cost of excavating each cut; and Table XXVI. gives the cost per cubic yard for each class of material. It should be remembered that wages were only 13½ cts. per hour for laborers, that two mules and a driver were 30 cts. an hour. To the costs given in the tables 6 per cent. should be added for superintendance and general expenses. The "average haul" was measured on the profile from center of gravity of cut to center of gravity of fill.

TABLE XXV.

					Cu. Yds.			
Case.	Explosives.	Labor and Foremen.	Teams, etc.	Total.	Earth.	Loose Rock.	Solid Rock.	Total.
I.	$96	$982	$404	$1,482	2,491	2,281	1,644	6,416
II.	$11	$167	$46	$224	717	517	.....	1,234
III.	$266	$1,619	$273	$2,158	2,619	2,831	2,435	7,867
IV.	$34	$278	$36	$348	.....	347	502	849
V.	$50	$359	$112	$521	1,953	596	.....	2,549
VI.	$19	$189	$36	$244	1,295	408	607	2,310

VII.	$185	$1,086	$196	$1,467	903	1,664	1,169	3,736
VIII.	$1,003	$2,483	$454	$3,940	176	177	6,568	6,921
IX.	$322	$1,659	$48	$2,029	.....	815	5,960	6,775
X.	$397	$1,924	$346	$2,667	2,043	1,504	2,934	6,481
XI.	$222	$883	$225	$1,330	764	969	2,089	3,822
XII.	$1,090	$4,978	$777	$6,845	868	1,117	11,676	13,661
XIII.	$23	$327	....	$350	400	600	......	1,000

TABLE XXVI.

Case.	Cost in Cents per Cu. Yd. Earth.	Loose Rock.	Solid Rock.	Average Haul, Ft.	Method of Hauling.	Time of Year.
I.	11.0	21.9	43.1	922	D	Nov., Jan., Apr. to July.
II.	12.8	25.6	....	150	D	Summer.
III.	12.3	24.6	46.8	350	C	Aug. to Feb.
IV.	....	26.7	50.8	200	D	Aug.
V.	16.6	33.2	....	225	D	Apr. and May.
VI.	5.5	11.0	21.1	425	D	Aug.
VII. ...	17.0	34.0	64.0	450	D	Winter and spring.
VIII. ..	15.5	31.0	58.7	300	C and D	Aug. to Dec.
IX.	....	16.5	31.8	50	W	Jan. to Mar.
X.	16.5	33.0	62.5	475	C and D	Aug. to July.
XI.	12.5	25.0	47.5	250	....	Summer.
XII. ...	14.4	28.8	58.4	610	C	Aug. to Aug.
XIII. ..	21.8	43.7	....	50	W	
Average	13.3	26.7	50.6	...	....	

NOTE.—The months given in the last column are inclusive. In the next to last column the letters denote as follows: D means by dump carts; C, dump cars; C and D, both dump carts and cars for about equal length of time; W, wheelbarrows.

The materials encountered in these 13 cuts were as follows:

Case I. Shale and slate ledges across part of cut, large and small sandstone boulders, stiff clay and earth. Work was stopped in wet weather on account of slides.

Case II. Disintegrated rock, sandstone boulders, fire clay and earth.

Case III. Slate rock in masses, shale, sandstone boulders and earth. A thorough cut first made, then borrowed from the sides. One mule pulled two cars.

Case IV. This work consisted in reducing a slope that was ¼ to 1, making it 1 to 1. Sandstone and rotten sandstone; the latter pulverized on being shot.

Case V. This was a borrow pit of disintegrated sandstone and average earth. Some of this sandstone should have been classified as solid rock.

Case VI. Slate in masses, fire clay, fire clay shale and debris from old slides consisting of boulders and earth. One foreman handled two gangs.

Case VII. This was a borrow pit, material being same as in Case VI.

Case VIII. Solid gray sandstone (seamy) and a small amount of earth and loose rock at each end. Thorough cut.

Case IX. This was a borrow pit and a continuation of the sandstone ledges of Case VIII.

Cases X. and XI. Mountain debris, consisting of sandstone boulders (large and small), fire clay, cemented gravel and earth, material always wet and heavy to shovel; slides were frequent in winter.

Case XII. Half of this cut was solid sandstone. At the other end was slate, fire clay shale, disintegrated shale, clay and large sandstone boulders. The cut was 27 ft. deep at deepest point. In the winter the shale slid on the fire clay shale, bringing down several thousand yards of slides into the cut. For five months (December to April) the shovelers stood in mud and water above their ankles. Material would frequently run out of the cars on the way to the dump.

Case XIII. This was a trench 6 ft. deep, 20 ft. wide on top, 15 ft. wide on bottom, dug to carry a creek.

It is not possible in all cases to compare the cost of one cut with another, but from the figures given several deductions can be made. In Cases X. and XI. the materials excavated were similar. There was a difference in the length of the haul, but the great difference in the cost of the work can be attributed to the time of the year that the excavations were made. One was worked entirely during the summer, while the other was worked during both summer and winter. If both cuts had been made in the summer months their costs should have been approximately the same, the slight difference in cost of haul counting in favor of the cut in Case XI. A valuable lesson can be learned from Cases VIII., IX. and XII. In these the greater part of the material was sandstone, being classified as solid rock. In two cases the work consisted of thorough cuts and one a side hill borrow. In VIII. the cost of explosives was a little more than $1,000, while with nearly twice the yardage in XII. the cost of the explosives was not $100 more. The difference in the cost of the teams in the two cases is readily accounted for when the lengths of the average hauls are compared. The great cost of powder in Case VIII. can to a great extent be attributed to the waste of incompetent foremen. The majority of boulders were "mud capped" instead of being "blocked," or having the charge of dynamite placed under the rock. In all, seven different foremen worked in this cut, four of whom were discharged as incompetent; but the damage was then done and the money in part lost. All thorough cuts were shot with Judson powder, as the rock is broken better than when black powder is used, and much "blocking" of boulders is prevented. Better results can always be obtained with Judson, except where it is desired to waste, when black powder should be used. In Case VI. Judson powder pulverized all the solid rock, so that no "blocking" was needed, as all the material was easily worked by hand. In Case IX., as it was very important that none of the material should be wasted, Judson powder was used, and less than $50 was spent in breaking up boulders. The cost of explosives in the entire borrow was only 4¾ cts. per cu. yd. of rock moved. In comparing all the cost items of this case with Cases VIII. and XII. the striking difference in the cost of side hill work and thorough cuts can be seen.

It was found necessary to widen and lower to grade certain cuts that had been left incompleted by another contractor. The cost of such skimming work is high, as is well shown in Table XXVII.:

TABLE XXVII.

			Cost per Cu. Yd.					
Case.	Cu. Yds.	Haul, Ft.	Explosives.	Labor.	Foreman.	Hauling.	Miscl.	Total.
XIV.	626	225	$0.098	$0.384	$0.108	$0.093	$0.001	$0.684
XV.	484	150	.108	.488	.122	.081	.001	.800
XVI.	415	325	.095	.334	.116	.091	.024	.660

Case XIV. was a breast 10 ft. deep on the side of a cut in hard blue sandstone.

Case XV. was the widening of both slopes of a cut in red sandstone and the excavation of the bottom.

Case XVI. was the excavation of the bottom of a cut in blue sandstone; at no place was the face excavation more than 5 ft. deep; 1½-yd. dump cars were used.

In cases XIV. to XVI. laborers received $1.50 per 10 hrs., and foremen $3 to $3.50; one driver and two mules on two carts were rated at $3.50 a day; powder was $1.20 a keg, and dynamite (40 per cent.) was 10 cts. per lb.

Cost of Blasting Boulders.—I am also indebted to Mr. Hauer for the following data on the cost of blasting "loose rock" in railway cuts. So far as I know these are the only records in print on this subject.

The boulders were red sandstone and blue sandstone, the latter being the harder and tougher of the two, and were all broken to sizes that could be loaded by hand. With wages of hand drillers and sledgers at $1.50 for 10 hrs., and 40 per cent. dynamite at 10 to 13 cts. per lb., the costs were as follows:

Sledging a 12½-cu. ft. blue sandstone boulder to small sizes took 1 man 5 mins. and 2 men 2 mins., at a cost of 4.9 cts. per cu. yd. Sledging a 16 cu. ft. blue sandstone boulder took 1 man 3 mins., or 1¼ cts. per cu. yd. Sledging a 12-cu. ft. blue sandstone boulder took 2 men 8 mins., or 9 cts. per cu. yd. Sledging a 7½-cu. ft. blue sandstone boulder took 1 man 3 mins., or 2.7 cts. per cu. yd. Sledging a 27-cu. ft. red sandstone boulder took 3 men 10 mins., or 7½ cts. per cu. yd. Sledging a 23-cu. ft. red sandstone boulder took 3 men 5 mins., or 4.1 cts. per cu. yd. Sledging an 18-cu. ft red sandstone boulder took 1 man 4 mins., or 1½ cts. per cu. yd. The average cost of breaking up small boulders with sledges was 5 cts. per cu. yd. The cost of mud capping 17 boulders is given in the following table, in which the first four boulders were red sandstone, and the rest were blue sandstone:

TABLE XXVIII.

Reference	Size of Boulder, Cu. Yds.	Labor, Cts.	Dynamite, Cts.	Cap and Fuse, Cts.	Cost per Cu. Yd., Cts.
1	17.0	30	$2.10	9	14.7
2	1.6	8	20	2	18
3	1.0	8	15	2	25
4	¾	8	10	2	27
5	1.0	8 11	15 10	2 2	48
6	1.2	8	20	2	25
7	1.4	8	25	2	25
8	0.7	8 11	10 15	2 2	68½
9	0.33	8	7½	2	52½
10	1.3	11	20	2	25½
11	0.45	11	15	2	62.2
12	0.5	11	15	2	56
13	0.5	11	10	2	46
14	0.5	10	7½	2	39
15	1.7	10	17½	2	17.3
16	1.0	10	7½	2	19½
17	1.0	10	12½	2	24½

Boulders No. 5 and 8 were blasted twice. In addition to the cost of mudcapping given in the table, the following was the cost per cu. yd. of sledging boulders: No. 5, 3 cts.; No. 6, 5 cts.; No. 7, 4 cts.; No. 10, 6 cts.; No. 15, 2 cts. The average cost of breaking up these 17 boulders by mudcapping was 36 cts. per cu. yd.

The cost of blockholing five boulders is given in the following table, all boulders except No. 1 being blue sandstone:

TABLE XXIX.

Reference Number.	Size, Cu. Yds.	Depth Drilled, Ft.	Cost of Drilling, Cts.	Dynamite, Cts.	Caps and Fuse, Cts.	Labor, Cts.	Total, Cts.	Total per Cu. Yd., Cts.
1	13	3	105	20	2	18	145	11
2	3	1	40	10	2	8	60	20
3	4½	1½	52	12	2	10	76	17
4	2½	1	30	8	2	5	45	18
5	2	2/3	20	10	2	4	36	18

The labor item in the seventh column includes the cost of lost time in leaving the pit during the blasting. Boulders No. 1 and 3 were fired alone, making this labor item greater than in the other cases when several boulders were fired at one time.

The cost of undermining and blasting sandstone boulders was as follows: A 50-cu. yd. boulder was broken up by undermining it, charging with black powder, and tamping with a large quantity of sand around it, the total cost being 25 cts. per cu. yd., distributed as follows:

4 kegs of powder, at $1.64	$6.56
2½ lbs. 40 per cent. dynamite, at 16 cts.	.40
Cap and fuse	.04
Labor	5.50
Total	$12.50

The labor force was 1 foreman at 30 cts. an hour, 2 laborers at 10 cts. an hour and 1 boy at 5 cts. an hour. Five other smaller boulders were undermined and blasted at costs given in the following table:

TABLE XXX.

Reference Number.	Size, Cu. Yds.	Labor, Cts.	Dynamite, Cts.	Cap and Fuse, Cts.	Total per Cu. Yds., Cts.
1	0.9	10	5½	2	19.4
2	1.5	13	16½	2	21
3	1.25	9	8	2	15.2
4	1.0	9	5	2	16
5	2.5	9	8	2	8.4

Boulders No. 1 and 2 were red sandstone, and laborers were paid 13½ cts. an hour; 40 per cent. dynamite, 11 cts. per lb. Boulders No. 3, 4 and 5 were blue sandstone; laborers were paid 15 cts. an hour; dynamite, 10 cts. per lb.

Comparing the average costs of breaking up sandstone boulders by the four different methods we have:

By sledging	5 cts. per cu. yd.
By mudcapping	36 " " " "
By blockholing	17 " " " "
By undermining	16 " " " "

In none of the cases did the men know that they were being timed, so that the costs may be assumed as fair averages. It is evident that mudcapping should be used only in steam shovel work through cuts where boulders must be broken up quickly so as not to delay the shovel. To illustrate how expensive it is to mudcap, one more example will serve. In a cut (approach to a tunnel) 1,054 cu. yds. of solid sandstone were excavated, 36 kegs of black powder costing $51 being used. In this same cut there were 512 cu. yds. of loose rock (boulders) which were broken by mudcapping, requiring 750 lbs. of 40 per cent. dynamite, costing $90.

For moving boulders short distances to a steam shovel, to a derrick or even to the dump, a stone sled (or boat or "lizard") should be used. Where many boulders are to be moved to the dump use three sleds, and several chains for each team. Have a dump crew and a loading crew, and thus, with the extra sleds, keep the team moving. Boulders from ¼ to 1 cu. yd. can thus be moved a short distance cheaper than by blasting and loading into wagons. By having a number of rollers at the dump, the dump crew can work the sled along on the rollers, using bars and levers, until it tilts up at the edge of the dump and discharges its load.

For loading large scattered boulders into wagons use a three-leg (or tripod) derrick with a Triplex block. A "lifter" or "devil" is a simple, handy, device for loading 3 or 4 cu. ft boulders by hand. The "lifter" is simply a stretcher made of two 3-in. round poles 8 ft. long, with 2-in. plank, 2½ ft. long, nailed across so as to make a platform on which to carry the boulder. Several men can thus carry a ½-cu. yd. boulder. Where a few large boulders are to be loaded upon a wagon, take off the two wheels on one side of the wagon and skid the boulders up into the wagon; then use levers to raise the wagon axles, and replace the wheels.

Where many boulders are to be loaded, a derrick car may be used. A car provided with an A-frame at the front end, a hoisting engine, and light jack arms, will lift 5-ton boulders. Such a car will cost about $1,000 and the engine will cost another $1,000.

When a steam shovel is used to load large boulders or loose rock that will not pass through the dipper, use chains. A chain should have a hook at each end, and one hook to fasten the boulder and the other to hook into a chain fastened to the dipper arm or to a chain on one of the dipper teeth. Two small boulders, ½ to 1 cu. yd. each, may be chained out at one time. For large boulders, 1 to 3 cu. yds., use a chain with a pair of grab hooks and keep one or more men busy drilling dog holes in the boulders for the grab hooks to bite into. With a 65-ton shovel it is possible to load a 4 cu. yd. boulder.

Summary.—The two cost items that the inexperienced man should seek first to inform himself upon, are: (1) The number of feet of hole drilled per cubic yard in different kinds of rock; and (2) the number of pounds of explosive required per cu. yd. under varying conditions. In Table XXXI. I have given a summary of these items as applying to open cut work discussed in this book; the table does not apply to trenching, tunneling or other narrow work. Two examples are given for sandstones and two for shales, such as occur in the coal measures of Pennsylvania. In a thorough cut on railroad work, we have conditions that approach trench work, requiring more feet of hole and more powder than in open side cuts; hence the differences between Examples 5 and 6, 7, and 8. It will be observed that the large amount of drilling in Example 2 is due to the shallowness of the face or lift, and in Examples 9 to 12 it is due to the toughness of the rock.

I shall greatly appreciate further contributions of similar data from my readers, for use in future editions. The greater the number of records, such as those in Table XXXI. the better will readers be able to judge the range and the average for each class of rock.

TABLE XXXI.

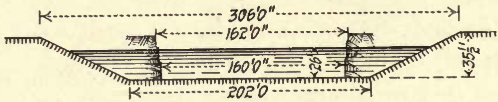
		Per Cu. Yd.				
Example.	Depth of Lift, Ft.	Ft. of Hole.	Lbs. of Black Powder.	Lbs. of Dynamite.	Grade of Dynamite.	Kind of Rock.
1	12	0.40	...	0.75	40%	Limestone, Chicago Canal.
2	6	1.00	...	0.7	40%	" for crushing.
3	20	....	...	0.37	50%	" for cement.
4	15	0.43	...	0.26	50%	" (holes sprung).
5	20	0.1	1.0	0.1	40%	Sandstone, side cut.
6	20	0.2	2.0	0.2	40%	" thorough cut.
7	24	0.08	0.7	0.03	40%	Shale, soft, side cut.
8	24	0.20	1.5	0.10	40%	" hard, thorough cut.
9	16	1.36	...	0.20	60%	Granite, for rubble.
10	12	1.33	...	0.60	40%	Gneiss, New York City.
11	14	0.63	...	0.50	40%	" " " "
12	12	1.7	...	0.67	40%	Syenite, Treadwell mine.
13	12½	0.32	...	0.44	52%	Magnetic iron ore.
14	14	0.35	...	0.20	75%	Trap, seamy.
15	16	1.0	...	0.70	40%	" massive.

CHAPTER XIII.

METHODS AND COSTS ON THE CHICAGO DRAINAGE CANAL.

General Conditions.—The illustrations in this chapter originally appeared in *Engineering News.* For cost data I am particularly indebted to Mr. W. G. Potter, to *Engineering News* and to *Engineering Record.*

No excavation work of such magnitude as the Chicago Drainage Canal has ever been so fully described in print; and, what is more noteworthy, no such complete record of cost has ever been published before. In view of these facts, and because such a variety of machines were used in excavating this canal, I have thought it wise to devote one chapter to this great work.



Chicago Main Drainage Channel.

Fig. 33.

The rock section of the Chicago Canal is 160 ft. wide at the base, and has vertical sides 36 ft. high. The rock was excavated in three 12-ft. lifts, channeling machines being used to cut perfectly smooth walls. The excavation was classified as solid rock and "glacial drift"; the latter including all material other than solid ledge rock. The solid rock was limestone (Niagara period), occurring in horizontal strata. The lower lift is said to have been almost twice as hard to drill and channel as the upper lift. The contractors were required to base their bids on the results obtained by borings. Mr. L. E. Cooley, the first chief engineer, intended to sink a number of test pits, but was overruled by the first Board of Trustees, which acted under the usual cents wise and dollars foolish policy of political "boards."

The borings gave an entirely misleading idea of the character of the glacial drift, and failed to indicate that much of it was an exceedingly tough hardpan of cemented gravel, and of gravel boulders mixed with clay. Several contracting firms were ruined by the subsequent action of the board in refusing to release them or to reclassify the material.

The average contract price on the "glacial drift" excavation was 30 cts. per cu. yd. and 78 cts. per cu. yd. on the solid rock. There were about 26,000,000 cu. yds. of "glacial drift" and 12,300,000 cu. yds. of rock. The prices for excavation included all bailing and draining. The contracts contained an excellent clause that required that the work done each month should be not less than such a proportion of the whole work as one month was of the total number of months agreed upon for completion of the work.

Excavating Very Tough Clay.—After removing the upper layer of prairie soil, an exceedingly tough or "indurated clay" was encountered that required blasting. Boulders were found scattered through the clay, and in some cases in such quantities as to make a regular hardpan, of which I shall speak later. On one section, 14 men were kept busy drilling and blasting for one Bucyrus shovel having an output of only 350 cu. yds. per 10-hr. shift. The shovel took out a swath 12 to 14 ft. deep, and blasting holes were put down 16 to 18 ft. deep, sprung with dynamite and charged with Judson powder. On another section Barnhart AA shovels averaged 520 cu. yds. per 10-hr. shift for seven months. As high as 600 cu. yds. was averaged by a Bucyrus shovel on another section, showing that the toughness of the material varied considerably. For data as to the shovel output and cost in this clay, the reader is referred to my book on earthwork.

Excavating Hardpan.—The worst hardpan encountered in this canal work is shown in Fig. 34. It consisted of boulders and stones cemented together so that a vertical face was left after blasting. The only way that this material could be loosened economically was by means of large charges of dynamite fired at the rear of small tunnels, as shown. The contractors who at first bid 28 cts. on this work subsequently secured 50 cts. a cu. yd. on a re-letting. On Section 4 of the canal there were four steam shovels; one 70-ton Osgood, one 60-ton Bucyrus and two 45-ton Bucyrus. According to the reports of the canal engineers, for October

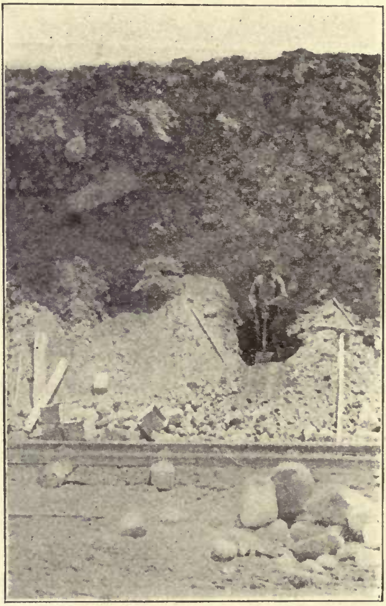


Fig. 34.

1894, the Osgood shovel worked 27 ten-hour shifts, averaging 406 cu. yds. per shift. The three Bucyrus shovels averaged 760 cu. yds. for the 60-ton shovel, 458 and 480 cu. yds. for the two 45-ton shovels. The general daily average for the season was 493 cu. yds. per shovel. The good record made by the smaller shovels as compared with the larger shovels was largely due to the fact that they were given the easiest material to handle, so that no conclusions can be drawn as to relative efficiencies of the respective shovels.

This remark will also hold true of most of the data that follow.

The material was loaded in 3-cu. yd. Peteler dump cars, hauled on a down grade in trains of two cars by two horses to the foot of the incline and hoisted up the incline by a Lidgerwood 12½ x 16-in. double drum engine, or by a 13 x

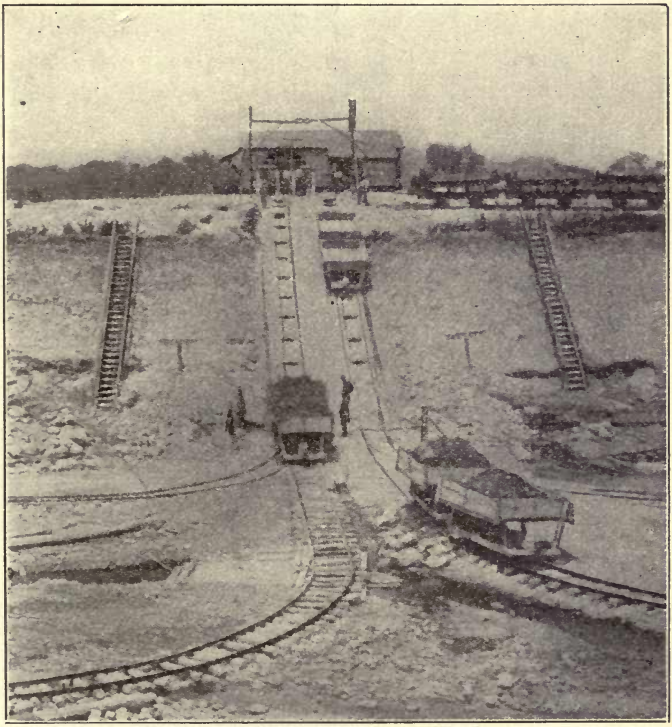


Fig. 35.

16-in. double drum Webster, Camp and Lane engine. Fig. 35 shows one of these inclines. It was found that 750 ft. was the limit of economic haul from the shovel to the foot of each incline. Hence the glacial drift was taken out of the canal for a stretch 1,500 ft. long before moving the incline. Fig. 36 shows the track arrangement.

For Section 3 the track layout is shown in Fig. 37. A Mundy 60-H. P. hoisting engine hauled two 3-cu. yd. cars at a time up the incline, and a team of horses then hauled the cars to the dump. A team can haul 4 to 6 empty cars

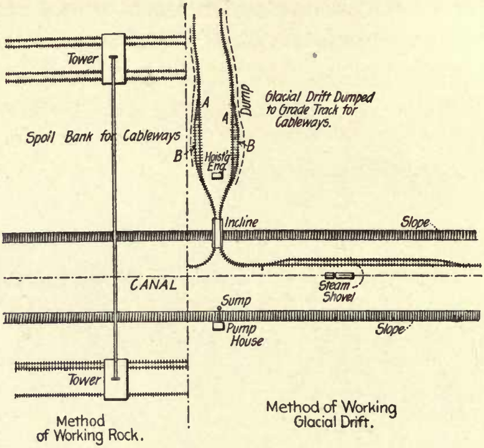


Fig. 36.

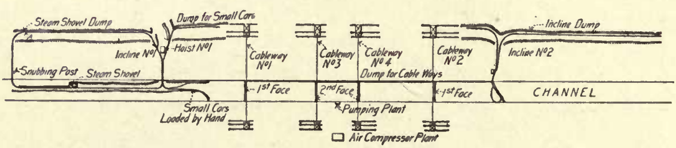


Fig. 37.

back. A Victor shovel was used on this work and its output by months was as follows:

Month.	Shifts.	Cu. yds. per shift.
Sept., 1894	21	362
Oct. "	24	316
Nov. "	18	333
Dec. "	25	392
Jan., 1895	17	235
May "	22	296
June "	21.6	380
July "	21	330

Incline and Tipple.—On Section 1 an incline with tipple was used for handling rock, and this was the only instance where this device was used for other material than earth. The incline is of steel, with the track rising at a 30° slope. There is a trestle approach to the tipple consisting of short king post spans supported on trestle bents, which in turn rest on greased skids so that they may be slid along. Fig. 38 shows the arrangement of the inclines and track system for the rock work, also the two derricks carrying the air-hoists used in loading large rocks into the cars. Fig. 39

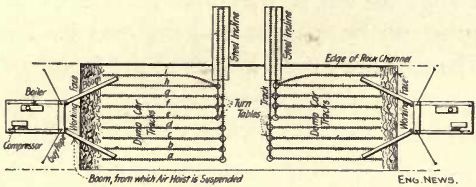


Fig. 38.

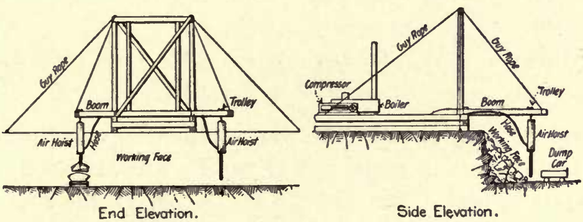


Fig. 39.

shows a derrick and two air hoists attached to two horizontal booms that swing right or left so as to cover the face of the quarry. The air hoist may be moved back or forward along the boom. Each hoist has a 12-in. cylinder 8 ft. long that is supplied with air from the compressor on the framework as shown. The compressor which has an 18 x 24-in. piston also supplies air to the drills. Six men operate the air hoists on each face changing from one hoist to the other, so that after loading the large stones on the right side, they move to the left hoist, leaving the right side free for men to load the small stones by hand into the cars. Two sets of chains and grab hooks were used with each hoist, two men handling each set of grabs. While the hoist is traveling along the boom to the car one pair of chainmen are hooking on to a rock at the face, and, while the empty hoist is returning, the other two chainmen are unhooking from the rock on the car and returning. On man runs the hoist and one tagman swings the boom and pulls the hoist back and forth. The cars when loaded are hauled by mules to the turn tables, shown in Fig. 38, where they are transferred to the incline track, hauled up by a cable and dumped by the automatic tipple. The cars are of steel, open at the front end, and there is one car to each of the nine tracks.

The output of the two incline conveyors was as follows:

	No. 10-hr. shifts.	Cu. yds. solid rock per shift.
May	38	166
June	67	160
July	91	164
Aug.	97	187

Cost by Lidgerwood Cableway.—Nineteen cableways were used on the canal, spans 550 to 725 ft. with traveling towers 73 to 93 ft. high. Mr. Charles H. Locher, of the contracting firm of Mason, Hoge & Co., invented an aerial dump, by means of which the skips could be dumped automatically. This device insured the success of the cableway. Fig. 40 shows the traveling towers. There are 5 cables. The main cable for the carriageway is 2¼ ins. diam. and has a sag of 5 ft. per 100 ft. The hauling cable and the hoisting cable are each ¾ in.; the button cable, for distributing and supporting the fall rope carriers is ⅝ in., as is also the dumping cable for dumping the load. The life of the main cable was 50,000 to 80,000 cu. yds. of solid rock, or 30,000 to 50,000 trips, or 100 to 160 days. A 70-H. P. boiler and a

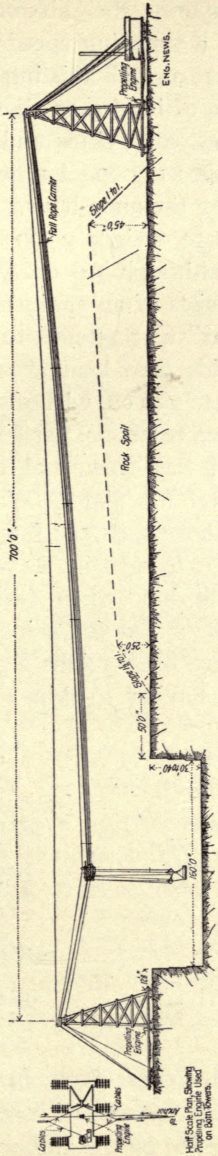

Eng. News.
Propelling Engine
Fall Rope Carrier
Slope 1 to 1.
45'0"
Rock Spoil
700'0"
25'0"
50'0"
160'0"
Propelling Engine
Cables
Cables
Propelling Engine
Anchor
Half Scale Plan, Showing Propelling Engine Used on Both Towers.

Fig. 40.

10 x 12-in. engine operate the cableway, giving a hoisting speed of 250 ft. and a traveling speed of 1,000 ft. per min. For shifting the towers a small hoisting engine on each tower car operates a system of blocks, as shown in Fig. 40. The total weight of cables, cars, skips and all is about 450,000 lbs., and the cost about $14,000. Large stones (6 to 8 tons) are chained out during the noon hour. The skip is 2 x 7 x 7 ft. of ¼-in. boiler plate, weighs 2,300 lbs. and holds 1.9 cu. yds. of solid rock, although 1.6 cu. yds. was the average load. The force consists of an engineman, a fireman, a signalman and a "rigger" who attends to oiling and changing worn out parts, beside men loading skips. The efficiency has ranged from 300 to 450 cu. yds. of solid rock per 10 hrs., handled at a cost of 28 to 30 cts. per cu. yd., including loading skips, pay of cableway crew, coal, oil, repairs and maintenance, but no rental for plant. Wages for 10-hr. shift were as follows: Laborers, $1.50; foremen, $3; engineman, $2.75; fireman, $1.80; towerman, $2.70. The fireman and towerman worked 12 hrs. when two 10-hr. shifts were worked each 24-hr. day. The following is the record for the month of March, 1895, on Sections 2 and 4 for four cableways, each cableway having 10 skips:

No. of cableway	1	2	3	4
No. 10-hr. shifts	49	35	52	49
Total cu. yds. rock	12,633	8,632	16,162	14,535
No. skip-load by day shift	5,111	5,327	5,435	4,369
No. skip-load by night shift	4,087	1,201	5,467	4,468
Cu. yds. solid rock per skip	1.44	1.32	1.48	1.65
Cu. yds. solid rock per shift	258	247	311	297
No. of laborers	27	27	32	32
No. of foremen	2	2	2	2
Total hours labor	12,861	9,608	17,075	15,227
Cu. yds. rock loaded per man per shift	9.82	8.98	9.46	9.54
Tons coal per shift	1.83	1.83	2.28	2.28

The contractors, McArthur Bros., furnished the foregoing data, also a table of percentages of cost, from which the following has been compiled by the author;

Percentages of Cost.

	Labor (2/3).	Supplies (1/3).	Total (3/3).	Assuming 50 Cts. per Cu. Yd., Cost per Cu. Yd. in Cts.
Drilling	22	10	18	9.0
Explosives	3	58	21	10.5
Loading	46	2	31	15.5
Conveying	15	20	17	8.0
Channeling	4	3	4	2.0
Pumping	4	7	5	2.5
Supt. and gen'l labor	6	..	4	2.0
Total......	100	100	100	50.0

While the contractors did not care to give out the cost in cents for each item, the author has deduced (algebraically from the data upon which the above table is based) that on one section of the canal the total labor cost two-thirds and the total supplies one-third of the cost of rock excavation, while on another section the total labor was 60 per cent. and the total supplies 40 per cent. of the grand total cost. During the months of May, June and July these same cableways on Sections 2 and 4 averaged 340 cu. yds. each per 10-hr. shift.

On Section 3 the output of four cableways, as given by the Supt. of Construction, was as follows:

	No. 10-hr. shifts.	Cu. yds. per shift.
Sept., 1894	126	294
Oct. "	140	267
Nov. "	153	230
Dec. "	140	305
Jan., 1895	173	161
May "	193	306
June "	181	308
July "	185	254

The contractors, Gilman & Co., gave the following as the output for May, which agrees very closely with the output given by the superintendent of construction: The average output working two 10-hr. shifts daily was 305.3 cu. yds. per shift per cableway, skips averaging 1.6 cu. yds. each; one cableway averaged as high as 346 cu. yds. and as low as 284 cu. yds. Assuming 36 laborers loading the 9 skips at each face (under one foreman), the average output per man per 10-hr. shift was 8.5 cu. yds. for the month of May. No delays are counted out, unless the men are actually laid off without pay; these delays for repairs and due to slight accidents were probably not less than 10 to 15 per cent. of the working time where two shifts were worked. Three Rand drills worked on each cableway face, receiving air from an 18 x 30-in. duplex compressor. A single row of holes across the canal was exploded at each blast; about 500 lbs. of dynamite per day being used on each face. Drilling cost 6 cts.; dynamite and blasting, 9 cts.; channeling, 7 cts., and dumping, 2 cts.; total, 24 cts. per cu. yd. Assuming 36 laborers, at $1.50, loading skips, 2 foremen at $3 and cable force at $7, 2 tons coal for cableway at $2, we have a total of $71 per 10-hr. shift for loading and conveying, say, 270 cu. yds., or 27 cts. per cu. yd., which, added to the 24 cts. above given, makes 51 cts. per cu. yd. exclusive of plant installation, plant rental and office expenses.

On Section 6 four cableways were used, and according to the contractors, Mason, Hoge & Co., the output was as follows for December, 1894, to March, 1895:

	No. of Shifts.	Length, Shift, Hrs.	Delay for Repairs, Hrs.	Delay for Skips, Hrs.	Cu. Yds. per Skip per Cableway.	Cu. Yds. per Skip.	Cu. Yds. per Man.
Dec.	82	9	37	138	333	1.30	9.5
Jan.	99	9	62	197	338	1.43	9.62
Feb.	70	9	28	102	293	1.33	8.37
Mar.	92	10	91	140	345	1.50	9.86

The small size of skip load is due to the large size of the pieces. It will be seen from the above tabulation that from

2 to 3 hrs. were lost through delays during each 10-hr. shift for the four cableways, or an average of nearly 40 mins. each 10-hr. shift for each cableway. Other records bear out this ratio, as on Section 7 where for one cableway the average delay per 10-hr. shift was 1.08 hrs. for a 3 mos. run, and on Section 8 a similar average was 1.12 hrs. lost per shift. In these cases cableways were not working night shifts, so that the conditions were favorable. The Lidgerwood Manufacturing Co. calls attention to the fact that the delays are due to two items: (1) Delay for repairs to cableway; (2) delay due to water in the pit, absence of rock ready to load, and the like. The first item, they claim, is generally about one-quarter of the entire delay, and they enumerate such items as the following: Door ring leaked during the night and boiler had to be refilled; bolt broke in dumping device; main cable sheave broke; dump line broke; dump line and hoist line tangled; box of tower sheave broke; button line broke. In this way nearly 13 hrs. were lost one month, while 16 hrs. more were lost waiting for stone on Section 7, where due to these delays and others the average output for the month was 403 cu. yds. per shift worked, or 476 cu. yds. per 10 hrs. actually run; and the month following the output was 462 cu. yds. per shift worked, or nearly 500 cu. yds. per 10 hrs. actually run.

On Section 7 nine skips and about 35 men were worked on a face. About 1½ tons of coal, costing $2 per ton, and 25 cts. worth of oil were consumed per shift. According to official report the average output of one cableway for several months was 332 cu. yds. per 10-hr. shift, but the contractors give an average output of 394 cu. yds. per 10-hr. shift for three months (Jan., Feb., Mar., 1895), the average skip load being 1.87 cu. yds., and the average 10-hr. output of each man loading skips being 10.1 cu. yds., assuming 35 men to have been the loading force. As a matter of fact, they worked only 9-hr. shifts, but for sake of uniformity the output has been reduced to a 10-hr. basis. The average lost time was 13 per cent. in Jan., 12 per cent. in Feb. and 6 per cent. in March.

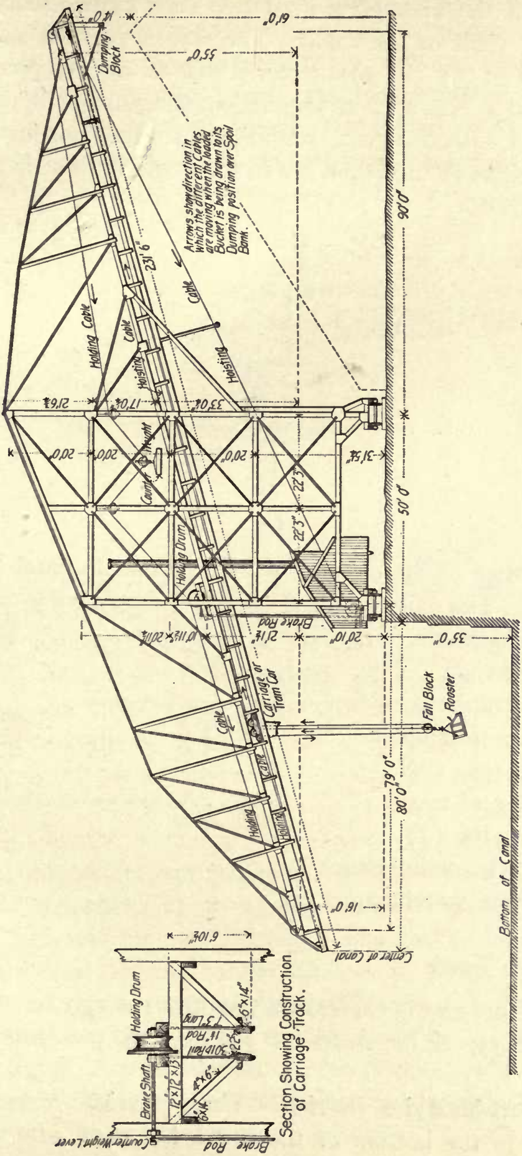
On Section 8 a row of 20 holes is drilled 8 ft. back from the face, and 8 ft. apart, each hole being loaded with 15 lbs. of 40 per cent. dynamite, breaking 28 cu. yds. of solid rock. As a consequence of this close spacing of holes, the rock was loosened in smaller sizes, and each skip load was correspondingly increased. Nine skips and 40 men loading were employed on each cableway. Locher, Harder & Williamson have furnished the following records for their two cableways for the first three months of 1895:

	No. Skips, Both Cableways.	Total Cu. Yds.	Total Delays, Hrs.	Total Shifts.	Cu. Yds. Output per Hr.
Jan.	10,485	17,475	80	37*	52.5
Feb.	6,869	14,809	63	30*	51.9
Mar.	8,180	13,491	64	30½	44.2

The January record on this section is excellent and shows what may be done where the plant is well handled and the rock well broken up, so as to reduce the delays arising from handling large rocks with grab hooks. The average skip load, it will be seen, was 1.8 cu. yds. It should also be noted that if the average force of loaders did not exceed 40 men, each man loaded nearly 11½ cu. yds. per 9-hr. shift.

Hullett-McMyler Cantilever Crane, or Conveyor.—This machine resembles the Brown cantilever crane, but instead of spanning the canal it reaches only to the center. Indeed, as first made it was designed to overhang the canal bank but 10 ft., and was then used to receive its skip load from a McMyler locomotive crane running on a track in the bottom of the canal; but this arrangement did not prove sufficiently efficient. Fig. 41 shows the general design of the Hullett-McMyler cantilever crane in its final form; and Fig.

* Jan., all 9-hr. shifts; Feb., half 9-hr., half 10-hr. shifts.



Dumping Block
Arrows show direction in which the different Cables are moving when the loaded Bucket is being drawn to its Dumping position over Spoil Bank.
231' 6"
Holding Cable
Hoisting Cable
Cable
Hoisting
Counter Weight
Holding Drum
Brake Rod
Carriage or Tram Car
Fall Block
Rooster
Holding
Hoisting
Cable
90' 0"
50' 0"
79' 0"
80' 0"
61' 0"
55' 0"
35' 0"
16' 0"
22' 3"
Center of Canal
Bottom of Canal
Section Showing Construction of Carriage Track.
Holding Drum
Brake Shaft
Counter Weight Lever
Brake Rod
6' 10½"

Fig. 41.

42 shows the arrangement of two such cranes working on opposite sides of the canal. The skip is of steel and has a capacity of 3.7 cu. yds. water measure, or 1⅝ cu. yds. of solid rock. A 9 x 12-in. engine, working under 80 lbs. pressure and with 200 revs. per min., does the hoisting. The total weight of the crane is 110 tons, and its cost is given at about $9,000.

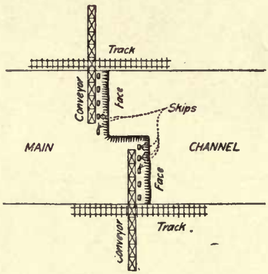


Fig. 42.

Only two of these cranes were used on the canal (on Section 7). The daily (10-hr.) expense of operating each was:

1 engineer	$2.50
1 fireman	1.50
Machinist service	1.00
Superintendence	.75
1¼ tons coal	2.50
Oil and waste	.25
Repairs (?)	.50
Track maintenance	1.50
Night watchman	.50
Total	$11.00

The two cantilever cranes handled 168,470 cu. yds. solid rock in 337 10-hr. shifts, or 250 cu. yds. per shift per machine.

Hullett-McMyler Derrick.—These derricks were at first placed in the bottom of the canal, but were afterward enlarged and placed on the berm, Fig. 43. No locomotive crane had ever before been made with so long a boom. The derrick handles a skip weighing 2,400 lbs., making, with its full load of 1⅝ cu. yds. solid rock, 3½ tons loaded. The machine was designed to handle safely a load of 10,000 lbs. The derrick weighs 95 tons, and its cost is given at $15,000.

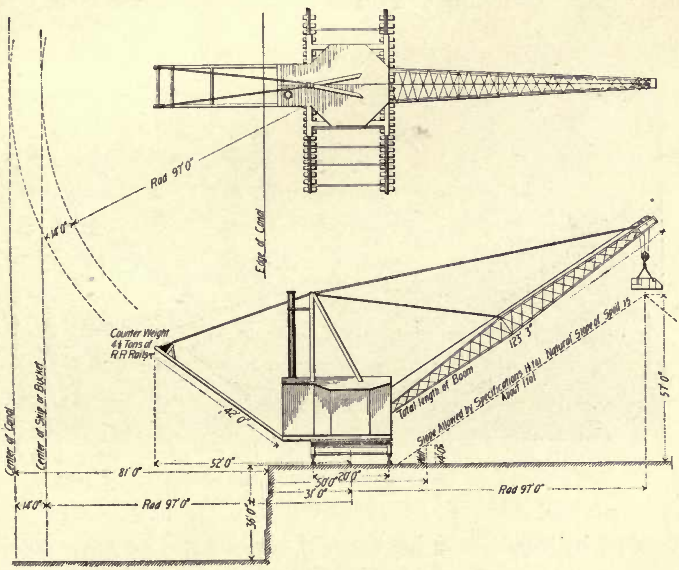


Fig. 43.

A cranesman, a fireman, a man to trip skips, 3 laborers hooking and unloading skips, 25 loaders and a foreman constitute the crew for each derrick. The cost of operation is practically the same as for the Hullett-McMyler conveyor.

In the pit 3 men were busy hooking and unhooking skips of which there were 5 or 6 to each machine. The highest daily output of which record is given was made Mar. 18, 1895, when 605 skips, or 980 cu. yds., of solid rock were conveyed in 10 hrs. by the two machines. To load this rock there were 59 laborers in the pit, so that each man handled 16.6 cu. yds. that day. All told, these two conveyors moved 279,300 cu. yds. in 492 10-hr. shifts, averaging 568 cu. yds. (284 cu. yds. each) per shift.

Geraldine Double-Boom Derrick.—Four of these derricks were used on the canal, the general appearance of each being shown in Fig. 44. The derrick revolves on a turntable, having a rack 28 ft. diam. in which two pinions mesh and are

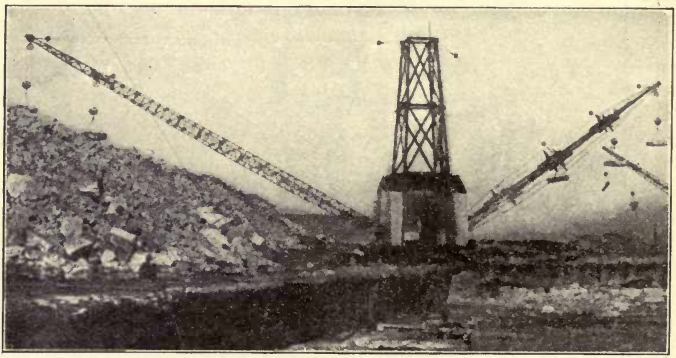


Fig. 44.

rotated by two 8 x 8-in. vertical engines. The apex of the tower is 113 ft. above the track. The booms are 155 to 164 ft. long, and each boom carries two fall blocks for lifting the skips. While one boom overhangs the canal the other overhangs the spoil bank. As soon as two skips are hooked on in the pit the engineman begins to raise them, and at the same time to swing the derrick. The skips upon reaching a certain height are dumped automatically. Hoisting is then stopped, while the opposite skips, which have been lowered, are unhooked and loaded skips hooked on. Twelve skips holding 2 cu. yds. each are used. The cost of operation is:

50 laborers at $1.50	$75.00
2 foremen	6.00
1 engineman	2.50
1 fireman	1.50
2 trackmen	3.00
2 tons coal	4.00
Oil, water, watchman and supt.	5.00
Total	$97.00

The output of four derricks for 5 mos. ending July, 1895, was 300 cu. yds. solid rock per derrick per 10-hr. shift, which would indicate that there were many delays, since if 50 men were engaged in loading the 300 cu. yds. the average per man must have been only 6 cu. yds. The best average for any one of these derricks for one month of this time was 439 cu. yds. per 10-hr. shift.

The Brown Cantilever Crane.—The cantilever crane manufactured by the Brown Hoisting & Conveying Co., of Cleveland, Ohio, had been used for many years for handling iron ore, coal, etc., along the Great Lakes and elsewhere, so that its success on the Chicago Canal might have been regarded as a foregone conclusion. However, the output of this device and its general efficiency far exceeded expectations. Altogether eleven of these cranes were used on the canal, and after the first year a monthly output of 15,000 to 16,000 cu. yds., or 600 cu. yds. per 10-hr. shift per crane, was attained; in fact, for a week one crane handled 892 cu. yds. per 10-hr. shift, or 4,845 cu. yds. Fig. 45 shows the general design of one of these cantilever cranes. The cantilever trusses have a slope of 12½° and are 355 ft. from end to end. A carriage or trolley travels along the track on the lower chord of the truss, the hoisting power being a 10½ x 12-in. engine and a 120-H. P. boiler. The skip can be dumped automatically at any point of its 343 ft. travel. The weight of the entire machine is 150 tons, and its cost about $28,000. Each skip has a capacity of 75 cu. ft. water measure, and carries

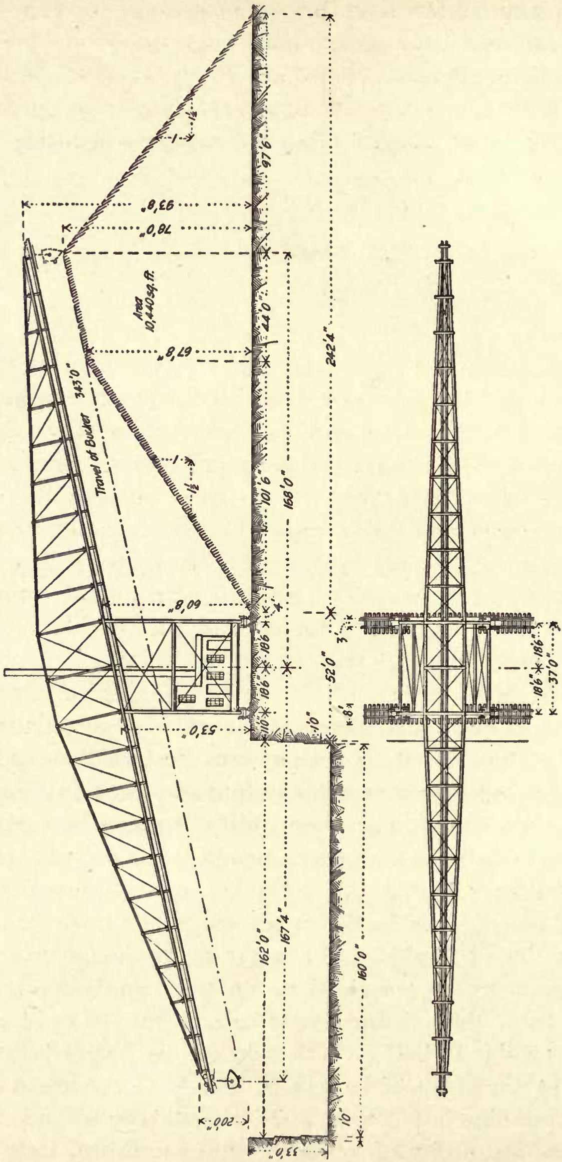

97'6"
93'8"
78'0"
Area 10,440 sq. ft.
24'0"
67'8"
242'4"
Travel of Bucket 343'0"
101'6"
168'0"
60'8"
18'6"
52'0"
53'0"
162'0"
167'4"
160'0"
20'0"
33'0"
37'0"

Fig. 45.

1.5 to 1.7 cu. yds. of solid rock. The average traveling speed of the skip is 150 ft. per min., although the maximum speed is 400 ft. per min. On Section 10 the contractors, E. D. Smith & Co., have furnished the following record of output for the month of March, 1895, for one crane working in the lower lift, using 9 skips:

Number of 10-hr. shifts	25
Output per shift, cu. yds.	520
Number of laborers on the face	48
Output per laborer per shift, cu. yds.	10.4
Av. skip load of solid rock, cu. yds.	1.6

There were two foremen, one crane engineer, one fireman, one oiler for crane and one towerman or operator; 2 to 2½ tons of coal were burned per shift. Mr. C. L. Harrison, the division engineer, gives the output of three cranes on this section for six months of 1895 (Feb. to July inclusive) as having been 205,900 cu. yds. in 421 shifts (10-hr.), or nearly 490 cu. yds. per crane per shift; during the month of April the average output for each of the three cranes was 542 cu. yds.

On Sections 11, 12 and 13 there were eight cantilever cranes operated by the manufacturers, who sublet the conveying of the loaded rock. These cranes handled 2,084,700 cu. yds. of solid rock in 3,609 shifts (10-hr.), or very nearly 500 cu. yds. per crane per 10-hr. shift. In the first 12 months (Feb., 1894, to Jan., 1895) the average output was 485 cu. yds. per crane per 10-hr. shift and 10.9 cu. yds. per man loading skips. The lost time per crane per 10-hr. shift for one month, of which record is given, was about 2 hrs., of which about ½ hr. was due to the crane and the other 1½ hrs. to the failure of the contractors to have stone ready to load, which is probably a fair average. In Dec., 1895, these eight cranes working one 9-hr. shift a day averaged 60.5 cu. yds. per hour. For the 12 mos. each of these eight cranes averaged 23 shifts a month; in mid-winter working 9-hr.

shifts, and in mid-summer 11-hr. shifts, although the records have all been reduced to a 10-hr. shift unit.

The daily cost of operating each crane was as follows:

Engineman	$3.00
Fireman	2.50
Oiler	1.75
Operator	2.75
1 2-3 tons of coal at $1.75	3.00
Oil, water and waste (estimated)	.50
Laying track (estimated)	.50
Total	$14.00

It will be seen that the conveying cost about 3 cts. per cu. yd., not including plant rental. No other method of conveying equaled this in point of low cost of operation, and it should also be noted that track had to be laid along only one berm of the canal; but, on the other hand, the price charged for a cantilever crane was considerably in excess of that charged for any other machine. If $28,000 represents the price of each crane and 260,000 cu. yds. of rock were handled by each crane, obviously the contractor would be compelled to charge nearly 11 cts. per cu. yd. against the crane, unless he could be sure of a definite salvage price for the crane at the end of the job. Considerations of this kind are too often overlooked by engineers in estimating the unit cost of work.

Cost by Car Hoists.—For conveying rock in dump cars there were three methods used: No. 1, the fixed incline with spur tracks; No. 2, the natural incline with a loop track, and No. 3, the double-hoist and diagonal incline.

Method No. 1. This method was used on Sections 8, 11, 12, 13 and 15 of the canal. A number of spur tracks were laid in the pit up to the rock face, and a number of spurs were laid on the dump, no spur being over 150 ft. long, and all connecting by switches with the main track, which led up a fixed trestle incline. This incline had a slope of 30°, 40 ft. of it being in the canal prism and 85 ft. up on the berm.

A hoisting engine was located at the head of the incline. Single cars (of which there were 8), each holding 1 cu. yd. solid, were loaded by hand, hauled by a mule to the foot of the incline, hoisted by the engine and hauled by a mule to the dump. The average haul (one way) from pit to dump was about 600 ft. The pit force was 40 to 45 men loading, 1 water boy, 1 mule and driver, and 1 foreman. The dump

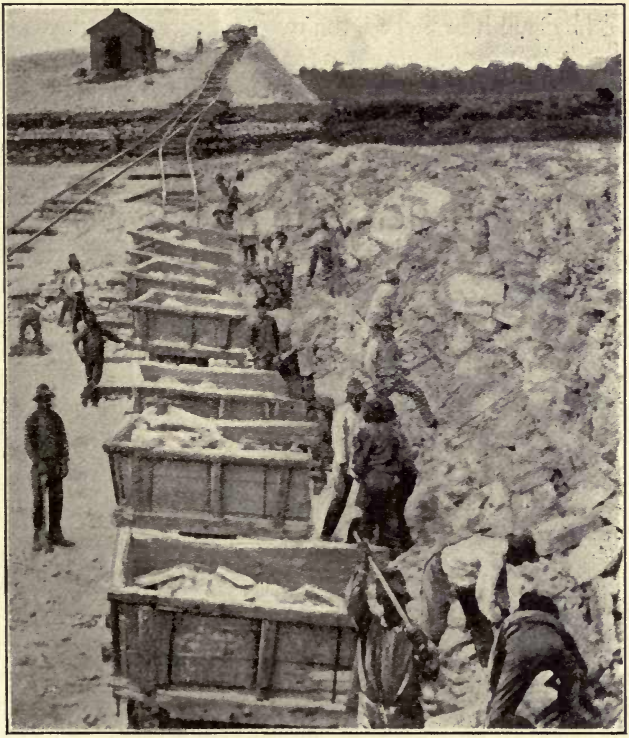


Fig. 46.

force was 1 hoist engineer, 1 hoist runner, 1 mule and driver, and 4 dumpmen. The incline was torn down whenever it had to be shifted.

Method No. 2. On Section 10 the upper lift was taken out by a car hoist method that differed from all others in that there was only one (3-ft.) gage track on the dump, a loop in the pit and practically no trestling on the incline. The empty cars descended by gravity around the loop, as shown in Fig. 46; and one incline served two working faces. The wedge of earth and rock which was left to support the track on the incline was removed in carts after the incline had served its purpose and had been moved. The main track on the incline was 150 ft. long; at a distance of about 75 ft. back from the canal it branched, the two tracks coming down over the berm and meeting at the far side of the canal, forming a loop 375 ft. long. This method of track arrangement gave a very short haul and the track was easily maintained. There were 3 trains of 4 side-dump cars each, two being loaded, while the third was at the dump; and each car held 1½ cu. yds. of solid rock. When a train was loaded the cable was attached and it was hauled up the incline. The pit force consisted of 35 loaders, 1 water boy, 1 cableman, 1 switchman and 1 foreman. The dump force was 1 hoist engineer and 4 or 5 laborers.

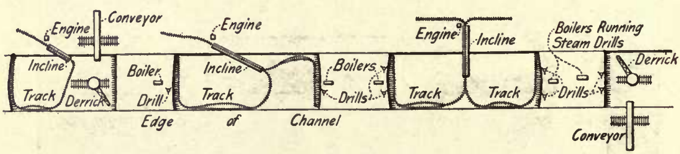


Fig. 47.

Method No. 3. Double Hoist and Diagonal Incline.—This was the only car hoist method used for all three lifts. A double track trestle incline entered the pit diagonally—not at right angles with the canal, like other inclines. The two working faces were also diagonal, making an angle of 30° with the canal, thus giving a longer loading line. The track layout is shown in Fig. 47. There were two parallel main tracks on the incline trestle, each 150 ft. long; then each track was 300 to 800 ft. long in the pit and 600 to 1,000 ft. on the bank. The average haul was about 700 ft. Two trains of 12 cars each were used on each face, so that there was no delay in loading. There were sidings in both pit and dump. Each car held ½ cu. yd. of solid rock. A double-drum hoisting engine handled two cables, one for each track. The pit force working two faces consisted of 75 loaders and edgers (1 sledger to 6 loaders), 3 teams, 2 water boys and . foremen. The dump force was 1 hoist engineer, 1 firemen, 3 teams and 10 dump men.

Summary of Costs.—The summary in Table XXXII. has been compiled by Mr. W. G. Potter. The wages for the different classes of workmen are given elsewhere in this chapter, common laborers in all cases receiving $1.50 for 10 hrs work, all delays of 1 hr. or more being docked. The tabulated costs do not include shop repairs, but do include field repairs. The drilling item appears not to include the cost of drill sharpening. Plant rental is not included either—a very important item where such expensive machines are used. Explosives include caps and dynamite, 12 cts. per lb. for the 40 per cent. dynamite being assumed to cover the cost of explosives. General expenses include superintendence, watchmen and incidentals. The three methods of car hoist, Nos, 1, 2 and 3, have been described in the pages just preceding.

TABLE XXXII.

COST IN CENTS PER CU. YD. (SOLID).

	Channeling.	Drilling.	Explosives.	General Expense.	Pumping.	Conveying.	Pit Force.	Dump Force.	Total.
Brown Cantilever	3.9	4.1	8.0	3.2	1.0	3.6	14.6	0.0	38.3
Lidgerwood Cableway.	3.7	3.8	8.4	2.7	1.0	3.6	15.6	0.0	38.8
Hullett-McMyler Derrick	3.9	4.0	7.4	2.5	1.8	5.3	18.3	0.0	43.2
Hullett Conveyor.....	4.1	3.7	8.5	3.8	1.2	6.2	21.4	0.0	48.9
Car Hoist No. 1......	3.7	3.9	9.1	2.7	0.8	3.1	24.8	5.1	53.1
Car Hoist No. 2......	3.9	3.6	8.9	3.2	0.9	1.2	22.9	2.3	47.1
Car Hoist No. 3......	4.0	5.0	10.7	3.1	1.2	1.2	26.4	4.8	56.5

TABLE XXXIII.

	Section.	Cu. Yds.	Cu. Yds. per 10 Hrs.	Cu. Yds. per Man in Pit per 10 Hrs.
Brown Cantilever	10	443,750	478	10.45
Lidgerwood Cableway	8	600,725	397	10.25
Hullett-McMyler Derrick	7	180,406	217	8.52
Hullett Cantilever	7	109,397	235	9.91
Hullett Conveyor............	9	178,839	335	6.85
Car Hoist No. 1.............	8	131,674	285	6.96
Car Hoist No. 2.............	10	60,341	269	6.98
Car Hoist No. 3.............	9	308,531	463	6.82
Double Boom Derrick........	14	324,880	282	8.22
St. Paul Derrick.............	14	63,700	153	8.22

Cost of Channeling.—Channelers had never been used on canal excavations before the building of the Chicago Canal. The object in using them on the canal was to secure smooth side walls that would offer little resistance to the flowing water. The limestone varied widely in hardness, the upper lift being as a rule much easier to channel than the lower lifts. There were three lifts of 12 ft. each, and the channel cut was offset 6 ins. at each lift. Naturally the lower lifts were somewhat shattered by the blasting of the upper lift, adding to the difficulty of channeling. The average weight of a channeler was 11,000 lbs.; it moved back and forth on a section of track 30 ft. long, striking 250 blows per min. The width of the channel cut by the bit was 2⅜ ins. at the top, decreasing ⅛ in. for each 2 ft. of depth. The speed of channeling was 130 to 200 sq. ft. per 10 hrs. on the upper lift, and about half as much on the second and third lifts.

There were 53 Sullivan and 33 Ingersoll channelers. The following are records of efficiency:

	Sullivan Channeler.			
	No. 101.		No. 111.	
	June, 1893.	Jan., 1894.	June, 1893.	Jan., 1894.
No. hours worked........	205	222	209	299
Sq. ft. channeled.........	2,783	1,401	3,076	1,683
Av. sq. ft. per hour......	18.4	6.3	14.2	5.6
Max. sq. ft. per day......	300[1]	126[3]	357[2]	96[3]
Min. sq. ft. per day......	90	18[3]	63[2]	12[3]

[1] 10-hr. day; [2] 11-hr. day; [3] 9-hr. day.

Labor troubles in June reduced the number of shifts worked, but the work was done on the top lift under the most favorable conditions. The work done in January was under the most unfavorable conditions, on the third lift, with ice and snow, frozen water pipes, etc.; and 5 night shifts were worked with No. 111. The time (hours worked) is that for which the men were paid, and includes all time lost for repairs and delays. The labor and fuel expense of running a channeler was as follows per day:

1 runner	$2.75
1 helper	1.50
1 fireman	1.75
Blacksmith and teams hauling drills	.70
Superintendence and machinist	1.30
1¼ tons coal	2.50
Total	$10.50

The Ingersoll-Sergeant Drill Co. has furnished the following record of work of 5 channelers for the month of May, 1894, working on the top lift: The average cut by each machine for 27 working days was 2,279 sq. ft., or 121 sq. ft. per day. As high as 221 sq. ft. were cut by one machine in a day, and the best of these 5 channelers averaged 148 sq. ft. a day for the month. The cost of operation was:

1 runner	$3.00
1 helper	1.50
1 fireman	1.75

Coal	$3.00
Blacksmith, machinist	1.50
121 sq. ft. at 8.9 cts.	$10.75

For an excavation 160 ft. wide the cost of channeling, at this rate, was slightly less than 3 cts. per cu. yd. These figures, however, apply only to firm limestone in the top lift, and 30 to 50 per cent. should be added to this cost for the lower lifts. Seven channelers on Section 9 working 7 mos. in 1894 averaged 94 sq. ft. each for the three 12-ft. lifts. Lewis gives the cost of channeling on Section 3 as being 7 cts. a cu. yd., which would be equivalent to about 20 cts. a sq. ft. *Engineering News* gives the average for the whole canal work as 8 to 10 sq. ft. an hour at a cost of 8 to 25 cts. a sq. ft., and places the average cost at about 17 cts. a sq. ft., or 6 cts. a cu. yd. of rock excavated.

Mr. W. G. Potter gives the following data for Jan. 1, 1894, to Feb. 1, 1895, on ten sections:

Section	Sq. ft.	Days (10-hr.).	Cost per sq. ft.	Sq. ft. per day.
No. 7	98,043	1,170	11.73 cts.	83.80
No. 8 (1st lift)..	42,000	300	7.50 "	140
No. 8	201,558	2,339	11.10 "	86.2
No. 9	182,167	1,992	12.16 "	91.4
No. 10	202,192	2,638	11.81 "	76.6

Time is actual working time, all delays of 1 hr. or more deducted. Shop repairs are not included. Wages per 10 hrs. were:

	No. 7.	No. 8.	No. 9.	No. 10.
Channelerman	$3.00	$3.25	$3.25	$2.75
Fireman	1.75	1.75	1.75	1.75
Laborer	1.50	1.50	1.50	1.50
Team (occasional)	3.50	3.50	3.50	3.50
Foreman (for entire sec.)......	2.75	2.75	2.75	....
Coal, oil and waste............	1.75	2.25	2.25	1.75

Cost of Drilling.—Compressed air was used on 9 out of 15 sections of the canal. The common installation was a large compressor located at the center of a section, delivering air at 80 to 90 lbs. pressure. An 8 or 10-in. main led to the canal berm and there branched, a 6 or 8-in. main going each way; 2-in. feed pipes, 175 to 230 ft. long, supplied three drills. The common size of drill was one with a 3¼ x 6½-in. cylinder, and a 2-ft. feed screw; + and ×-bits were used, the ×-bit being better. For a 12-ft. hole the starting bit was 2 ins. diam. According to W. G. Potter the expense per drill per 10-hr. shift was about as follows:

	Steam.	Air.
Drill runner	$2.00	$2.00
Drill helper	1.50	1.50
1/3 fireman	.50	(air) 1.50
Coal and oil	1.25	(oil) .10
Total	$5.25	$5.10

On Section 9 the daily average for 6 mos. was 82 ft. of hole per drill for each of 15 Ingersoll-Sergeant drills; each hole being 12 ft. deep. Holes were spaced 6 to 12 ft. apart; close spacing decreasing the cost of sledging, but increasing the cost of drilling.

Many grades of dynamite were tried, but finally 40 per cent. was found to be the best. The higher grades shattered the rock in the immediate vicinity of the holes, but threw down chunks that were too large to handle. The sticks of dynamite were 1½ x 6 ins. in size, weighing ¾ lb. each; and 10 to 25 sticks were charged in a hole, thus consuming 0.6 to 1.2 lbs. of 40 per cent. dynamite per cu. yd. Including fuse and caps the dynamite averaged about 12 cts. per lb.

Steam Shovel Output.—Steam shovels were not much used for loading rock, as their average output was small. On Section 15 a 20-ft. face was worked, and 5-cu. yd. cars were loaded by two Bucyrus (55-ton) shovels with broad, shallow dippers having a capacity of 2¼ cu. yds. Cars were hauled in trains of 10, one locomotive serving each shovel. The best record made by one shovel was 600 cu. yds. in 10 hrs. The combined output of these two shovels was as follows:

1895.	No. of 10-hr. Shifts.	Cu. Yds. per Shovel per Shift.	Total Cu. Yds., Solid Measure.
May	109	266	29.000
June	99	291	28,850
July	96	302	29,000
Aug.	102	310	31,800

The average per shovel per month was 14,700 cu. yds., working two shifts a day.

CHAPTER XIV.

COST OF TRENCHES AND SUBWAYS.

General Considerations.—Trenching in rock is a subject upon which practically nothing has ever been written. In consequence there is probably no class of rock work that is so often mismanaged; and, as a further consequence of the prevailing ignorance, engineers' estimates of cost are often far too low and occasionally as far too high. In city specifications for sewer trenching in rock it is customary to pay the contractor only for rock excavated within specified "neat lines." If he excavates beyond the "neat lines" he does so at his own expense. In sewer work the most common practice is to specify that payment will be made for a trench 12 ins. wider than the outside diameter of the sewer pipe, and 6 ins. deeper than the bottom of the pipe when the pipe is laid to grade. A specification should always name a minimum width of trench. Some specifications allow for a side batter of 3 ins. to the foot on each side; but unless the trench is to be deeper than can be drilled with one set up of the machine drills, and requires excavation in more than one lift or bench, I see no reason for prescribing a batter to the rock sides. The most rational specification that I have seen for general use in rock trenching is as follows: "All trenches in rock excavation will be estimated 2 ft. wider than the external diameter of the pipe and 6 ins. below the sewer grade."

Specifications vary so widely as to the "neat lines" that bidding prices for trench work under different engineers are very deceiving to any one who has not studied the particular specifications covering the work upon which the bids were made.

Different rocks vary greatly in the way the sides and bottom shear off upon blasting. The sides of trenches in soft rocks can be cut off clean when the blast holes are properly loaded; but tough granites, traps, etc., leave jagged walls, generally involving excavation beyond the "neat lines" specified. In excavating thin bedded, horizontally stratified rocks the drill holes seldom need to go much, if any, below the neat lines; that is, 6 ins. below the bottom of the pipe. But in excavating thick bedded and tough limestones and the like, it is generally necessary to drill 12 ins. below the bottom of the pipe. In tough granites, traps, etc., it is often necessary to drill at least 18 ins. below grade in order to leave no knobs or projections after blasting that would require breaking off with bull points and sledges. Obviously the shallower the trench the greater is the importance of making due allowance for this extra drilling. If a trench is only 3 ft. deep and it is necessary to drill 1 ft. below grade, then 33 per cent. must be added to the cost of drilling to grade in order to cover the cost of the extra drilling below grade; but if the trench is 10 ft. deep, then only 10 per cent. of extra drilling is required. I have known cases where engineers have lowered the grade about 6 ins. after a long stretch of rock trench had been completed, and have required the contractors to do this 6-in. skimming at the regular price per cubic yard! As a result, it has cost the contractor several times what he received for the work. In such cases the engineers have generally been ignorant of the actual cost of trench work; for otherwise they doubtless would have allowed an extra price.

Charging the Holes.—In tunneling, the explosive is most effective if placed in a pocket at the end of the drill hole. In narrow trench work, on the other hand, the explosive should be distributed all along the hole, leaving only enough length for the least possible amount of tamping. To one who gives the matter thought the reason is obvious, yet I have seen contractors actually "springing" holes in a hard limestone trench, and thus wasting much labor and powder.

Contractors often take out deep trenches in several benches, simply because they think it necessary to place all the charge together. As a matter of fact a trench 25 ft. deep, or as deep as the machine will drill, can be taken out in one lift. To do this the explosive is separated into several charges in the hole, tamping being placed between the charges. Where charges are separated in this manner firing should never be done with a fuse, but always with a battery. On page 204 I have described a method of charging alternate sticks of dynamite and wood plugs, the firing of the top stick sending all the others off. In a city this method could probably not be used with safety because of the danger attending such heavy blasting.

Use of a Blasting Mat.—For preventing accidents due to flying rocks all blasts in cities should be covered either with timbers or with a blasting mat. This should be done to avoid suits for damages, regardless of city ordinances. A blasting mat is readily made by weaving together old hemp ropes, 1½ in diam. or larger. To make such a mat, support two lengths of 1-in. gas pipe parallel with one another and as many feet apart as the width of the mat is to be. Fasten one end of the rope to one end of the pipe; carry the rope across and loop it over the other pipe; bring it back around the first pipe; and so on until a sufficient number of close parallel strands of the rope have been laid to make a mat as long as desired. Starting with another rope, weave it over and under, like the strands in a cane-seated chair, until a mat of criss-cross ropes is made. Such a mat, weighted down with a few heavy timbers, will effectually prevent small fragments from flying at the time of blasting. The mat and its ballast may be hurled into the air several feet, upon blasting; but it will serve its purpose by stopping the small pieces of rock which are so dangerous even where light blasts are fired. The mat should be laid directly upon the rock. Such a mat will save a great deal of labor involved in laying a grillage of timbers over a trench. It will also make it unnecessary for the blasters to stand far from the blast when firing.

Cost of Drilling and Blasting.—Next to tunneling there is no class of rock excavation requiring so much drilling per cubic yard as does trench excavation. In granites, if shallow holes are drilled by hand, the holes are frequently spaced not more than 1½ ft. apart. If in a very narrow trench 1½ ft. wide two holes are drilled in a row, one on each side of the trench, and if the rows are 1½ ft. apart, we have two holes drilled in a square 1½ ft. on a side; that is, for every 2¼ cu. ft. of rock we must drill 2 ft. of hole, or 24 ft. of drill hole per cu. yd. If the cost of drilling is 25 cts. a foot, we have $24 \times .25 = \$6$ per cu. yd. as the cost of drilling alone. It is seldom, however, that such narrow trenching is done. Trenches for small pipes are usually 2½ to 3 ft. wide; two holes are then drilled in a row, and rows are usually about 3 ft. apart. A trench 3 ft. wide with two holes in a row, and rows 3 ft. apart, requires 6 ft. of drilling per cubic yard. With drilling costing 50 cts. per ft., as it often does where hand drills are used in granite, the cost is then $3 per cu. yd. for drilling alone. Unless the job is too small to pay for installing a plant, hand drilling should never be used in trench work, because the drilling forms such a very large part of the cost.

In a trench 6 ft. wide in hard trap rock three holes were drilled in a row, one close to each side and one in the middle, and the rows were 3 ft. apart, thus requiring 4½ ft. of drill hole per cu. yd. of excavation. The drilling was done with steam drills at a cost of 30 cts. per lin. ft., for the holes were only 4½ ft. deep, the rock was hard, and the men slow, about 35 ft. being the day's work per drill. The contractor had to drill 1½ ft. below grade in this rock to insure having no projecting knobs of rock. While it cost $1.35 per cu. yd. to drill the 3½ ft. for which payment was made, to this must be added nearly 30 per cent., or $0.40 per cu. yd., to cover the cost of drilling the extra 1 ft. for which no payment was received, making the total cost of drilling \$1.75 per cu. yd. of pay material. About 2 lbs. of 40 per cent. dynamite were charged in each hole, making about 2.6 lbs. of dynamite per cu. yd. of pay material. The explosives thus added another \$0.40 per cu. yd., making a total of \$2.15 per cu. yd. for drilling and blasting.

In the same trap rock, where the trench was 8 ft. wide and 12 ft. deep, there were three holes in a row and rows were 4 ft. apart, requiring 2.53 ft. of hole per cu. yd. of pay excavation, plus 0.21 ft. of hole per cu. yd. of pay material to cover the cost of drilling the last 1 ft. of hole below the "neat line." Each drill averaged 45 ft. of hole in 10 hrs., and the cost was 23 cts. per ft. of hole; hence, $2.74 \times 0.23 =$ \$0.63 per cu. yd. was the cost of drilling. About 4 lbs. of 40 per cent. dynamite was charged in each hole, or 1.1 lbs. per cu. yd. of pay material, making the total cost 80 cts. per cu. yd. for drilling and blasting. A comparison of this cost of 80 cts. with the \$2.15 above given brings out strikingly the fact that each problem of trench work must be considered in detail by itself.

In a city where the contractor must shoot comparatively small shots in order to avoid accidents to buildings and suits for damages arising from "disturbing the peace," it is seldom possible to space the holes more than 3 or at most 4 ft. apart. In trenching in soft sandstone in Newark, N. J., where the trench was 14 ft. wide and 10 ft. deep, there were five holes in a row (the distance between holes being 3½ ft.) and rows were 4 ft. apart, making 2.4 ft. of hole per cu. yd. Each hole was charged with 4.12 lbs. of 40 per cent. dynamite, making practically 1 lb. per cu. yd. About half the dynamite is charged at the bottom of each hole, then tamping is put in, and the other half is charged up to about 2½ ft. below the mouth of the hole. Each steam drill averaged 90 ft. of hole per 10 hrs., making the cost of drilling 10 cts. per ft. of hole, or 24 cts. per cu. yd. Including the cost of dynamite and the placing of timbers over each blast, the cost of drilling and blasting was 40 cts. per cu. yd. This is probably as low a cost for breaking rock in trenching as can be counted upon under favorable conditions. In this rock there was no necessity of drilling below grade.

I am indebted to Mr. F. J. Winslow for the following data on trench work in Boston, Mass. House sewer trenches are never less than 3 ft. wide, and trenches for water pipe (16 ins. or less) are 2½ ft. wide. The rock is granite, and the drill holes are usually 3 ft. apart. On small jobs hammer drills are used, one man holding and two striking. For a hole 10 ft. deep the starting bit is 2½ ins. and the finishing bit is 1¼ ins. diam. A drilling gang of three men averages 8 to 10 ft. of hole in 10 hrs., although in soft rock 20 ft. may be drilled in 10 hrs. Forceite containing 75 per cent. nitroglycerin is commonly used, ½ to 3 sticks being charged in a hole. Force account records for granite trenching show that the average cost during the past 15 years has been $3.80 per cu. yd., including excavating and piling up the rock alongside the trench.

I am indebted to the Harrison Construction Co., of Newark, N. J., for the following information: In a sandstone trench about 6 ft. wide the holes were spaced about 3 ft. apart, thus requiring 4½ ft. of hole per cu. yd. In seamy rock, shallow holes 4 to 6 ft. deep were drilled, and from 2 to 3 sticks of 50 per cent. dynamite were charged, each stick being 1½ x 8 ins. This is equivalent to 0.55 lb. per cu. yd. Where the rock was solid, the holes were drilled 8 to 10 ft. deep and the dynamite charge doubled.

The cost of throwing rock out of shallow trenches or of loading it into buckets to be raised by the engine of a derrick, a locomotive crane or a cableway, is somewhat greater than the cost of handling rock in open cuts. A fair day's work for one man is 6 cu. yds. of rock loaded, when there is little sledging; but the output may be only 4 cu. yds. where there is a large amount of sledging to be done.

If cableways or derricks are used for hoisting the rock, bear in mind that they will be idle most of the time, for the drilling limits the output. With a given number of drills to a cableway, estimate the number of cubic yards of rock that the drills will break per day and divide this yardage into the daily cost of operating the derrick. Thus, in a trench 6 ft. wide, if the holes are 3 ft. apart, each cubic yard of rock requires 4½ ft. of hole, and each drill will break $13\ ^1/_3$ cu. yds. per day where 60 ft. of hole is a day's work. With four drills per cableway the daily output is $4 \times 13\ ^1/_3 = 53\ ^1/_3$ cu. yds. The cableway would be capable of handling several times this output were it not limited by the drilling. Notwithstanding that all this seems self evident, I have known more than one contractor to overlook the fact that the cost of handling rock from trenches is very much greater than in open cuts where holes are farther apart and where a few drills can keep a cableway busy. In my book on earthwork I have given in detail the cost of operating a cableway on trench work, and elsewhere in this book will be found the cost of hoisting with derricks.

Cost of N. Y. Subway Work.—Aside from some data on the cost of subway work given in my book on earthwork, nothing, so far as I know, has ever appeared in print on the actual cost of subway excavation. By observation and through the aid of an assistant I have secured reliable data relating to every item of cost on several typical sections of the New York Rapid Transit Ry., including excavation, concrete, steel construction, etc.; and it is astonishing to find how high the labor cost of the work has been. The high cost may be attributed to several causes. In the first place, the contractors were compelled to employ union labor, much of which was inefficient. In the second place the foremen on this work were, as a rule, paid such small salaries that the best class of foremen were not kept. In the third place excavation and other work in crowded city streets is obviously made difficult; the supporting of pipes, tracks, etc., adding greatly to the cost in certain parts of the city. In fact, in the lower part of New

York, where the material is all sand, I have found that 50 cts. per cu. yd. has been expended in shoring, bracing, etc. In the fourth place the light blasts required by city rules leave the tough mica schist in large chunks upon which much labor must be expended in gadding and sledging; for practically all the rock was broken to one or two-man size so that it could be hauled away in dump wagons.

The work that I am about to describe involved the excavation of about 125,000 cu. yds. of tough mica schist in the upper part of the city, where the streets are not crowded and where there were very few pipes to be supported. The width of the excavation was 41 ft., and the depth averaged about 30 ft. One trolley track ran along the center of the street and had to be supported the entire distance. This track supporting was accomplished at comparatively slight expense by using some ten second-hand railroad bridge trusses of 66 ft. span, which were moved forward as the work progressed. Five cableways, each having an average span of 400 ft., were used for hoisting the rock in self-righting buckets, which were dumped into patent dump wagons.

The average daily force employed was as follows:

	Rate per day.	Total.
4 foremen	$3.50	$14.00
80 laborers	1.50	120.00
10 drill runners	2.75	27.50
10 " helpers	1.50	15.00
2 blacksmiths	2.75	5.50
2 " helpers	1.50	3.00
5 hoisters	3.00	15.00
1 compressor man	4.00	4.00
1 fireman	2.00	2.00
2 timbermen	2.00	4.00
3 waterboys	.75	2.25
20 teams	4.50	90.00
Total per 8-hr. day		$302.25

The average output of this force was only 150 cu. yds. of rock per day!

	Wages per 8-hr. Shift.	Cost per Cu. Yd. Average of 30 Months.	Cost per Cu. Yd. Best Month.
Drill runners	$2.75	$0.174	$0.150
Drill helpers	1.50	.100	.082
Blacksmiths	2.75	.032	.025
Blacksmiths' helpers	1.50	.018	.012
Compressorman	4.00	.016	.014
Firemen	2.00	.012	.014
Hoist enginemen	3.00	.100	.051
Carpenters	3.50	.008	.000
Timbermen	2.00	.024	.000
Waterboys	0.75	.012	.010
Laborers	1.50	.785	.745
Foremen	3.50	.102	.095
Teams (with drivers)....	4.50	.620	.581
Total Wages.......		$2.002	$1.779
Cu. Yds. Excavated		125,000	7,600

To the foregoing must be added the cost of fuel, explosives, maintenance, interest and depreciation of plant, etc., as follows:

	Cost per cu. yd.
$^1/_{30}$ ton coke, at $4.50	$0.150
0.6 lb. 40 p. c. dynamite, at 12½ cts.	0.075
½ exploder, at 4 cts.	0.020
Drill repairs (est'd at 50 cts. a day per drill)	.034
Installing boiler and compressor	.014
Interest and depreciation (50 p. c.) of $7,000 boiler and compressor plant	.028
Ditto for $3,500 drilling plant	.014
Total supplies, etc.	$0.335
Add total wages	2.002
Total	$2.337

To this sum should be added 3 or 4 per cent. to cover general expenses, such as office rent, bookkeeping, night watchmen, insurance on laborers, etc., which would bring the grand total to nearly $2.40 per cu. yd. of rock excavated. It will be seen by the description of the work and by the comparatively low cost of timberwork that the expense of supporting pipes and tracks was unusually low for such a city as New York. On the other hand, the cost of drilling was exceedingly high, being 28 cts. per cu. yd. for wages alone, if we include the blacksmiths' wages and half the wages of the compressorman and his fireman. The drills should be charged with about half the cost of the fuel, which adds 7½ cts. more per cu. yd., making 35½ cts. per cu. yd. for drilling, not including some 3½ cts. for drill repairs (estimated) and 1½ cts. for interest and depreciation. Adding these two items we have a total of 40 cts. per cu. yd. chargeable to drilling alone, which is exceedingly high for an open cut of this width and depth. It is a striking fact that each drill broke less than 15 cu. yds. of rock per 8-hr. day. The inefficiency of the laborers is also well shown by their output of less than 2 cu. yds. per man per 8-hr. day. It is true that they had to do a great deal of gadding, sledging and hand drilling to break the rock ready to load into buckets; but any one who saw the men at work must have been impressed with their slowness. The output of only 30 cu. yds. per day per cableway shows how the cableway output was limited by the drilling. The high cost of hauling is also noteworthy, for the average haul was but little more than one mile.

While it was difficult to get union laborers to do a fair day's work, I think that if the contractors along the subway had in all cases employed civil or mining engineers of known experience in rock excavation, a great deal of money would have been saved.

CHAPTER XV.

SUBAQUEOUS EXCAVATION.

Cost of Rock Excavation in the Detroit River.—I am indebted to Mr. Chas. Y. Dixon, U. S. Assistant Engineer, for the following data of cost, which were originally compiled by Mr. Harry Hodgman, and which, so far as I know, are the most detailed and complete cost records of subaqueous excavation that have ever been published. In the *Michigan Engineer,* 1903, Mr. Hodgman gave a very complete description of this work upon which he has been continuously engaged since 1895. From Mr. Hodgman's article and from Mr. Dixon's letters to me, I have compiled the following: The work was done under three contracts, as follows: At Ballards Reef, the Buffalo Dredging Co.; at Lime Kiln Crossing, James B. Donnelly, of Buffalo, N. Y.; and at Amherstsburg Reach, M. Sullivan, of Detroit, Mich.:

At the mouth of the Detroit River it is usually necessary to drill and blast the material before it may be excavated. The drill boats in use are from 60 to 80 ft. long and from 25 to 30 ft. wide, and are held in position by four spuds, one at each corner. They are equipped with two or more Ingersoll steam drills supported on vertical frames having trucks to permit of the drills being moved horizontally along the edge of the boat. The drills are raised and lowered during the operation of drilling by hydraulic lifts. The boiler furnishes the steam for operating the drills, the pumps used in connection with the hydraulic lifts, the forge, the electric light plant and other machinery with which the ordinary drill boat is equipped. The drill boat usually serves the purpose of a machine shop where repairs are made to the entire dredging plant. It is always conveniently near to all parts of the work, and ordinary repairs are quickly made, the contractor usually providing a great variety of tools and machinery for use in cases of emergency.

The drill boats are usually operated day and night. The holes (about 2½ ins. in diam.) are made at the corners of 5-ft. squares to a depth of about 3 ft. below the required depth, at the rate of about 5 ft. per hour per drill. The amount of explosive used is about one pound of 60 per cent. dynamite per linear foot of drill hole. The holes are charged by inserting the sticks of dynamite with the exploder and battery wires attached into the bottom of a long pipe, the battery wires leading out through a slit in the side of the pipe. This pipe is lowered into the drilled hole, the dynamite shoved down with a long ram rod, and the pipe withdrawn, a wire spring clamped to the dynamite stick preventing its coming out of the hole. The wires are then attached to the battery and the dynamite exploded. During this operation the drill boat is not moved, nor does the work of operating the other drills cease except at the time of firing. On two occasions, however, the charge of dynamite came out of the hole, and was exploded directly underneath the boat, causing it to sink almost immediately. This may be attributed to carelessness, however, as before exploding the dynamite the battery wires should be drawn up until taut, indicating that it is in place. A quantity of dynamite is always kept conveniently near the work, but no more than one day's supply is kept at the drill boat, and this is stored in a small scow trailing off the down-stream end of the boat at a safe distance.

The dredges used in excavating the material are of the type known as the dipper dredge. They vary in length from 80 to 135 ft. and in width from 30 to 40 ft., and are held in position by three spuds (36 ins. square), two at the bow and one at the stern. The machinery for operating dredges varies greatly, the best recently-constructed dredges being equipped with machinery for raising the dredge on the forward spuds (known as pinning up) instead of by swinging the dipper as formerly. The dredge is moved forward in the cut by means of the dipper arm, and the width of the cut is usually from 15 to 20 ft. The capacity of the dredge dipper varies from 2 to 5 cu. yds. in rock work and from 4 to 7 cu. yds. for earth work. The amount of material removed by one dredge per hour varies from 20 to 100 cu. yds. in rock, and from 75 to 125 cu. yds. in earth. The best type of dredges, however, in soft earth and under favorable conditions are capable of removing from 250 to 300 cu. yds. per hour. The time delayed for repairs usually varies from one-fifth to one-third of the time actually worked.

After the entire width of the area to be improved has been worked over by the dredge, cut by cut, the derrick scow follows after to remove such loose pieces of rock as may have been left projecting above the required depth. The derrick scows are usually from 80 to 100 ft. long and from 20 to 25 ft. wide and they are equipped with an ordinary hoisting engine and derrick capable of lifting from 12 to 18 tons, and a complete diving outfit. When lifting the boulders, the derrick scow is pinned up and supported on two spuds, each about 1 ft. square. The material to be removed is found by means of an iron bar, about 30 ft. long, suspended from the side of the scow to the required depth. Any obstruction struck by this bar as the scow is swept over the improved area is removed by the derrick by means of a chain, which is placed in position by a diver.

After the area has been thus improved and cleared of obstructions an examination is made on the part of the United States to determine if the required depth has been secured. This examination consists in sweeping the entire area with bars suspended to the required depth. These bars (about 20 ft. long) are suspended by chains from a raft (100 ft.

long and 20 ft. wide) built of squared timbers, the raft being held in position by a rope leading to a head anchor and pulled back and forth by means of ropes leading to side anchors. Any obstruction found during this examination is removed by the derrick scow with diving outfit. During the progress of the work, as well as during this final examination, constant attention is paid to the water gage in order to allow for the fluctuations in the water surface. Following this examination and on the completion of the work required under the contract, the final survey is made, which survey consists in the taking of soundings at regular intervals as described above. On this final survey and on the original survey depend the estimate for final payment.

At Ballard's Reef the material excavated was limestone bedrock, clay, hardpan and boulders. Generally there was from one to two feet of loose material overlying the bedrock. About 50 per cent. of the material was hardpan and clay. The material within about 75 per cent. of the area improved required to be drilled and blasted before removal. In this material one drill boat, equipped with three drills and working double shifts, was used to provide work for one dredge.

At Lime Kiln Crossing the material was mainly limestone bedrock, with no overlying material. Within the entire area it was necessary to do drilling and blasting before dredging. In this material two drill boats (each equipped with three drills and working double shifts) were used to provide work for one dredge.

At Amherstburg Reach the material was limestone bedrock, clay and boulders. Generally there was from one to two feet of loose material overlying the bedrock. Within about 75 per cent. of this area it was necessary to do drilling and blasting before dredging. On this work two drill boats (each equipped with three drills and working double shifts) were used to provide work for two dredges continuously and for a part of one season three dredges.

TABLE XXXIV.

Dredging Detroit River, 1900-1903.

	Ballards Reef.	Lime Kiln Crossing.	Amherst-burg Reach.
Cu. yds. above 23 ft. depth (place measure)	74,143	101,072	98,332
Cu. yds., total (place measure)	135,548	121,707	209,821
Cu. yds., total (scow measure)	153,097	168,633	......
Area dredged, sq. yds	225,000	61,000	225,000
Average depth dredged, pay material	1 ft.	5 ft.	1.3 ft.
Average depth dredged, total exc	1.8 ft.	6 ft.	2.8 ft.
Average depth of water over material	21 ft.	18 ft.	20.7 ft.
Dredge hours, worked	7,248	3,945	9,021
Dredge hours, delayed	3,386	1,490	2,890
Dredge hours, total	10,634	5,435	12,911
Dredge months (12-hr. days)	34	17.4	39
Cost per month	$3,000	$3,200	$3,200
Total cost	$102,000	$55,720	$124,800
Cost per cu. yd. (place measure), pay material	$1.38	$0.55	$1.27
Cost per cu. yd. (place measure), total exc.	$0.75	$0.46	$0.60
Average cu. yds. per hr., working time	19.0	43.0	23.3
Average cu. yds. per hr., total time	12.8	22.4	16.2
Maximum cu. yds. per hr., soft material	150	250	200

TABLE XXXV.

Drilling.

		Ballards Reef.	Lime Kiln Crossing.	Amherstburg Reach.
Drill hours	Worked	24,442	37,746	38,441
	Delayed	982	1,278	......
	Total	25,424	39,024	38,441
Number of holes drilled		30,023	29,236	35,432
Number of feet drilled		191,850	240,591	181,421
Ft. per hr., actual work		7.9	6.4	4.7
Ft. per cu. yd., pay material		2.6	2.4	1.8
Ft. per cu. yd., total exc.		1.4	1.9	0.9
Distance between holes		5 ft.	5 ft.	5 ft.

Average depth of holes.....	6.2 ft.	8.2 ft.	5.1 ft.
Average depth of pay material	1.0 ft.	5.0 ft.	1.3 ft.
Percentage of drilling below pay depth	84.0%	37.5%	75.0%
Number of pounds of 60% dynamite	110,305	222,396	263,672
Lbs. per cu. yd., pay material.	0.5	2.2	2.7
Lbs. per cu. yd., total exc....	0.8	1.8	1.2
Total cost of drilling........	$59,235	$105,245	$96,470
Per cu. yd., pay material.....	$0.80	$1.04	$0.98
Per cu. yd., total exc........	$0.44	$0.865	$0.46
Per. lin. ft. drilled..........	$0.31	$0.44	$0.53
Per drill hour..............	$2.25	$2.69	$2.51

ITEMS OF COST.

Ballards Reef.

25,424 drill hours at $0.80	$20,340
110,305 lbs. dynamite at $0.15	16,545
2,450 tons of coal at $3.00	7,350
Repairs (approximate)..	5,000
Miscellaneous supplies .	5,000
Depreciation of plant...	5,000
Total	$59,235

Tug service included in dredging.

Lime Kiln Crossing.

39,024 drill hours at $0.80	$31,220
222,400 lbs. dynamite at $0.15	33,360
3,555 tons of coal at $3.00	10,665
Repairs (approximate).	5,000
Miscellaneous supplies .	5,000
Depreciation of plant...	5,000
Tug service, 30 mos. at $500	15,000
Total............	$105,245

Amherstburg Reach—

38,441 drill hours at 80 cts.	$30,750
263,372 lbs. dynamite at 15 cts.	39,500
3,740 tons of coal at $3	11,220
Repairs (approximate)	5,000
Miscellaneous supplies (approximate).....	5,000
Depreciation of plant (approximate)	5,000
Total	$96,470

Tug service included in dredging.

TABLE XXXVI.

Derrick Scows.

	Ballards Reef.	Lime Kiln Crossing.	Amherstburg Reach.
Cost	at $9.70 11.2 mos. $10,865 Tug service included in dredging.	13.0 mos. at $9.70 $12,610 Tug service included in dredging.	16.0 mos. at $9.70 $15,520 Tug service included in dredging.
Cost per sq. yd. of area improved	$0.0475	$0.22	$0.07
Cost. per cu. yd. of material removed by diver	$5.73	$2.84	$2.04

Summary of Cost.

Dredging	$102,000	$55,720	$124,800
Drilling	59,235	105,245	96,470
Derrick scows	10,865	12,610	15,520
Totals	$172,100	$173,575	$236,790
Cost per cu. yd. of pay material	$2.32	$1.718	$2.41
Cost per cu. yd. of total excavation	$1.27	$1.425	$1.04

Basis Upon Which Estimates of Cost Were Made.

Dredge Crew:	1 Captain at $125 per month	$125
	1 Runner at $90 per month	90
	1 Cranesman at $90 per month	90
	1 Fireman at $55 per month	55
	3 Deckhands at $40 per month	120
	1 Scowman at $40 per month	40
	1 Cook at $50 per month	50
	1 Watchman at $40 per month	40
	Total	$610

Tug Crew:	1 Captain at $115 per month	$115
	1 Engineman at $100 per month	100
	1 Fireman at $55 per month	55
	1 Deckhand at $40 per month	40
	Total	$310

Subsistence of 14 men per month	$150
210 tons of coal at $3	630
Repairs and supplies per month	500
Repairs at the end of season, per working month	500
Depreciation in value of plant per month of operation equals ⅛ of 10% of value of plant.	

DRILLING

3½ men per drill (at $2.50 per day of 11 hours) equals 80 cts. per drill per hour.

DERRICK SCOW

1 foreman at $90 per month	$90
1 engineman at $85 per month	85
1 diver, 25 days, at $10 per day	250
1 diver's helper, at $75 per month	75
6 deck-hands, at $50 per month	300
15 tons of coal at $3	45
Repairs and supplies per month	50
Depreciation in value of plant	75
Cost per month	$970

Cost of Harbor Excavation, Oswego, N. Y.—In *Engineering News,* Feb. 15, 1894, Mr. Wm. Pierson Judson gives the following data on rock excavation in the inner harbor of Oswego. N, Y. Over an area of 4,500 sq. yds. the rock had to be excavated to a depth of 15 ft. Over 70 per cent. of this area the rock had a face of 1 ft. or less, and over the rest the face was 2 ft. or less. The rock was gray wacke sandstone in horizontal strata 1 to 2 ft. thick, with seams in which the drill often jammed. The rock varied greatly in hardness; the drill often cutting 10 ft. with one sharpening, and at other times wearing dull in 1 ft. The rock excavated was 2,956 cu. yds., let to Hingston, Rogers & O'Brien at $2.75 per cu. yd., place measure. Work was begun June 30, 1893, with a very efficient plant.

The drill scow was 6½ x 26 x 82 ft., the bottom being of 8-in. oak, and the sides of 6-in. pine, and after a season's work showed no ill effects from blasts fired directly under it in 12 ft. of water. Its draft was 2½ ft. A deck house 14 x 72 ft. housed boiler, engine and blacksmith shop. On one side of this house was a 6½-ft. track carrying two drill frames each, one a separate truck that could be moved by two men operating a 5-ft. lever and ratchet engaging a 10-in. pinion on the truck shaft. Each drill frame carried a 5-in. Ingersoll drill, suspended from a 6-in. x 12-ft. hydraulic lift set vertically in the drill frame. The great value of this hydraulic hoist was that it could pull a drill loose instantly when stuck in a seam. These hydraulic hoists were operated by a duplex Blake pump with a 7½-in. steam cylinder, 4½-in. water cylinder and 10-in. stroke, working under 80 lbs. steam pressure. Steam for the drills, pump and 15-H. P. hoisting engine was supplied by a 30-H. P. boiler burning 1½ tons of coal in a working day of 22 hrs. There were two crews of six men each, and a blacksmith and helper with each crew.

Range marks 10 ft. apart made it possible to locate a drill within 1 ft. of any desired spot. A 4½-in. casing pipe was lowered and forced into the gravel overlying the rock. This pipe had a double T, 2 ft. above the bottom, to allow drill chips to escape. The pipe remained in position until the hole was drilled and charged. The drill steel is 26 ft. long, the upper 14 ft. being of 1½-in. machine steel; the next 10 ft. of 1¾-in. steel, and the lower 2 ft. of 2-in. octagon steel with a 3¼-in. square cross bit tempered in a saturated solution of equal parts of sal ammoniac, salt and alum.

Holes were drilled 2 to 4 ft. below grade, and spaced 5 ft.

apart in rows 5 ft. apart. The average depth of 1,000 holes was 5¼ ft. and the average time to drill each of these holes was 1 hr., although in rock free from seams a hole may be drilled in ½ hr., whereas in seamy rock 3 hrs. may be consumed. The maximum rate of penetration of the drill was 1 ft. in 4 mins. About 12 drills per 22 hrs. were sharpened. About 20 ft., or 240 lbs., of 2-in. octagon steel for 6,000 ft. of holes.

Dynamite (75 per cent.) in waterproof cases, 2½ x 18 ins., gave the best results. The cartridge is placed in an iron loading pipe, which hangs by a small tackle from the drill frame. When the bottom of the hole is reached, a plunger, which is within the loading pipe, is unclamped and forced steadily down upon the cartridge, while the loading pipe is slowly hoisted. The plunger and loading pipe are next raised through the casing; then the casing pipe itself is raised 4 or 5 ft. from the bottom; and in this position the charge is fired, one hole at a time, without moving the drill boat. From 2 to 6 lbs., average 2¾ lbs., of dynamite are fired in each hole. The entire time from the stopping of the drill to the firing, as just described, is 3 mins. in ordinary work, and often only 1½ mins. To shift the drill on its trucks 5 ft. lower the casing and start drilling a new hole takes about 2 mins. and can be done in 1 min. Fourteen holes 5 ft. apart can be fired from one setting of the drill boat. The rock is broken up into pieces of 1 to 2 cu. ft. each. Occasionally the dredge dipper brings up a rock too large to drop through the dipper. In such cases the dipper is rested on the dump scow, while a hole is drilled in the rock with a hand drill, loaded with ½ lb. of dynamite, tamped with cotton waste and fired without injury to the dipper.

The loading pipe, above mentioned, is worthy of description. Its lower end is a 2⅛-in. pipe 3 ft. long, with a ⅜-in. slot its full length. The upper end of this slotted pipe joins a 1-in. pipe 22 ft. long, within which works a ¾-in. plunger

27 ft. long, having a 2-in. head at its lower end resting on the cartridge. The leading wires from the cartridge pass out through the slot. A clamp at the upper end of the 1-in. pipe holds the plunger until ready to use. The device is all of iron and works perfectly. While the exact cost to the contractors is not known, the following estimate is based upon 1,000 cu. yds. of rock, for which 1,650 holes, aggregating 8,660 lin. ft., were drilled in 33 days of 22 hrs. each:

33 days' wages of drill crew, at $31	$1,023.00
4,000 lbs. of 75 p. c. dynamite, at 17 cts..	680.00
1,800 exploders at 3 cts.	54.00
49½ net tons of soft coal, at $3	148.50
42 1/3 gals. cylinder oil for drills, etc., at 30 cts.	12.70
55 gals, kerosene for lanterns, at 12 cts. ..	6.60
260 lbs. octagon steel, at 15 cts.	39.00
55 lbs. machine steel, at 4 cts.	2.20
General machine shop repair bill	34.00
Total, 1,000 cu. yds. ,at $2	$2,000.00

To dredge 1,000 cu. yds. required about 10 days' work of 10 hrs. each, costing, say, $500, making a total cost of $2.50 per cu. yd. of rock excavation, not including plant rental. While 1,000 cu. yds. were removed above grade, for which the contractor was paid, there was probably an equal amount of loose rock left below grade, for which, of course, no payment was made.

Cost of Excavating Black Tom Reef, N. Y.—In Farrow's Military Encyclopedia are given some valuable data on submarine rock excavation, from which I have abstracted the following relating to the excavation of Black Tom Reef in New York Harbor. Mr. W. L. Saunders was in charge of this work and designed apparatus which marked an epoch in submarine drilling and blasting. The work was begun May 2, 1881, and was completed in 344 actual working days (35 days were lost by storms and 26 in equipping scow).

The drilling plant consisted of three 5-in. Ingersoll drills mounted on a platform supported by spuds; a scow anchored alongside carried the boiler that furnished steam to the drills. The longest drill steel used was 28 ft. long, and the shortest, 16 ft.; the starting bit was 3¾ ins.; the finishing bit, 2½ ins.; and 9 ft. of hole were drilled on an average with each bit before sharpening. The drilling shift was 10 hrs. long, one shift a day; and 20.8 ft. of hole per drill per shift was the average drilled in the mica schist, not including the penetration of some 6 ft. of sand and gravel overlying the rock. Mr. Saunders invented an "ejector" which is a pipe surrounding the drill steel and through which water is forced to wash away the gravel and sand. 1,736 holes were drilled, 1,629 charged and 1,542 blasted; the average depth in rock being 10.17 ft. The distance between holes was 4 ft.; the area excavated, 32,100 sq. ft., and the rock removed, 5,136 cu. yds., place measure. The dynamite was 75 per cent., of which 20,461 lbs. were used; exploders, 1,844; drill steel, 395 lbs.; connecting wire, 77 lbs.; coal, 200 tons at $4.14; hose, $491; water, $500. For each cubic yard 3.44 ft. of hole were drilled, and 3.98 lbs. of dynamite used. The cost of the plant was:

Barge and equipment	$6,640
Two drill floats	9,082
Alterations, machinery, etc.	5,663
Total	$21,385

The cost of drilling and blasting was:

	Cost per cu. yd.
3.98 lbs. 75 p. c. dynamite	$1.84
1.22 oz. steel	.02
Coal and water	.25
Labor (payroll, $26.76 per day)	1.79
Repairs, plant	.31
" drills	.01
" ejector pipes	.05

Repairs, hose	$0.09
Wire and tape	.01
Total	$4.37

To this must be added the $1.95 per cu. yd. paid for dredging by contract.

Cost of Undermining Flood Rock.—This work is described in detail in Farrow's Military Encyclopedia, and while the method of undermining is not likely to be used again for harbor deepening, there are certain features that merit attention. Flood Rock was a 9-acre obstruction in New York Harbor, and under the direction of Gen. John Newton it was finally removed in 1885. Two shafts were sunk and 10 x 10-ft. drifts or galleries were run at right angles to one another, leaving pillars of rock 15 ft. square supporting a rock roof averaging 19 ft. thick, although in places it was only 10 ft. thick. In driving the drifts very small charges of rackarock were fired, one hole at a time, to insure safety from flooding through unexpected seams in the mica schist rock. Any seams encountered were plugged with cement. In driving the 10 x 10-ft. drifts, 6 lbs. of rackarock and 12 ft. of drill hole were required per cubic yard. The total drifting was 21,669 ft., or 80,232 cu. yds., requiring 480,000 lbs. of explosive. The pillars and roof (270,717 cu. yds.) were drilled and charged with 1.04 lbs. of explosive per cu. yd. of rock, requiring 0.42 ft. of drill holes per cubic yard. The final charge in the pillars and roof was 240,400 lbs. of rackarock and 43,300 lbs. of No. 1 dynamite, in 11,789 drill holes in the roof and 772 drill holes in the pillars, or a total of 113,120 ft. of drill holes. The cost of the work and explosives required in preparing for the final blast was $2.69 per cu. yd. of total excavation; and the rackarock used in the final blast cost $106,510. The loading of the final charge (283,730 lbs.) of explosives was done by 20 men working 8 to 12 hrs. a day for 70 days. Experiments had shown that a 10-lb. charge of No. 1 dynamite under water would explode another charge in a copper cartridge 27 ft. away, so that no electric connecting wires were needed between drill holes. The rackarock was loaded in thin (.005 in.) copper cartridges, which were soldered with a solder that was melted with wet steam. The main charge in each hole was rackarock, but the last cartridge in each hole was No. 1 dynamite, which was allowed to project 6 ins. outside of the hole. Every 25 ft. apart along the drifts were placed cartridges (2½ x 24 ins.) of No. 1 dynamite packed solid in a thin copper shell; and directly above each of these cartridges was a rigid brass cartridge (2 x 8 ins.) containing No. 1 dynamite packed loosely and an electric exploder. The mine was flooded with water and fired. There was no loud report and the concussion was comparatively slight.

A contract was let for dredging the rock at $3.19 per cu. yd.; but pending the award of the contract a derrick scow was used and removed 15 to 30 tons daily at a cost slightly less than the subsequent contract price. Large blocks were chained by divers. Later the contractors raised 120 tons a day, using two large grapple dredges. It is apparent from these meagre data that the rock was broken in large chunks which were dredged with great difficulty.

The Derby Tubular Drill Bit.—Lieut. Geo. McC. Derby (now Major of Engineers, U. S. A.) invented a drill bit that was used in drilling on the Flood Rock work, and it proved so greatly superior to the X-bits that I regard it as worthy of special description. Maj. Derby writes me that he patented the drill bit in 1885 and sold the patent rights to the Rand Drill Co., which, for reasons unknown to him, has never placed it upon the market. The drill steel was hollow, as was also the bit which was provided with six points or teeth. The bits were sharpened very much like the bits used in the plug drills made by the C. H. Shaw Pneumatic Tool Co., of Denver, Colo. Each bit was only 2 to 6 ins. long and fastened to the end of the hollow wrought iron drill rod with a steel pin or expanding copper ring. This saved steel and saved transporting long, heavy drill rods to and from the blacksmith shop. This bit was used with the ordinary percussive air drill, and, in drilling, a small core was formed which broke up under a slight blow on the drill rod. The chips were washed out of the hole by a current of water that was forced down through the hollow drill rod. The water was introduced into the hollow drill rod, either through the rotating bar or through a sleeve surrounding the piston rod which was lengthened for this purpose; the first method being the best. Maj. Derby informs me that the coarse chips of rock broken off by the bit are washed out whole, instead of being reduced to dust, which saves power and time in drilling a hole of given depth. This fact is well shown by the following comparative records: Experiments were conducted for several months of actual work, during which time 39,119 ft. of hole were drilled with X-bits and 39,200 ft. with the Derby tubular bit. The holes were about 9 ft. deep, and Rand "Little Giant" drills were used. As a result of this competition it was found that the tubular bit drilled 51½ per cent. faster than the X-bit, and that the diameter of the bottom of the hole was 25 per cent. greater than with the X-bit, which in itself is a decided advantage. Using a starter X-bit of 3¼ ins., the bottom of a 10-ft. hole was 2 ins. diam.; but with the tubular bit the bottom was 2½ ins. diam. Moreover the tubular bit made a perfectly round hole, which lessens the chances of a bit's sticking. It seems to me that the greater speed of drilling with the tubular bit was due to the use of a jet of water to wash out the chips, which also accounts for the fact that the bit does not wear so rapidly. Whatever the reason, the record of excellence of the tubular bit is well worthy of serious consideration by all who are interested in economic drilling.

Drilling and Dredging Way's Reef.—Way's Reef, New York Harbor, was removed in 1874. The crew was 35 men, consisting of 1 draftsman, 2 divers, 3 carpenters, 1 engineer, 8 drillers, 1 blaster, 1 blacksmith, 2 blacksmith helpers, 12 sailors, 2 firemen, 1 timekeeper and 1 tide gage keeper. This crew worked two shifts on the U. S. steam drilling scow. At first the starting bits were 3½ ins., but later it was found that by using a 5½-in. bit more explosive could be placed in a hole resulting in breaking the rock up much better, even with comparatively wide spacing of the holes, which is a point well worth remembering. The average depth of drill hole was 8.13 ft., but only 6½ ft. of hole were averaged per drill per 8-hr. shift. About 3,030 cu. yds. of mica-schist were excavated, 15,308 lbs. of nitro-glycerin being used. The cost of dredging and dumping the rock was $4.29 per cu. yd., the dredge averaging 35 cu. yds. per day. The total cost of this excavation was $18.26 per cu. yd. The work was done by day labor for the Government, and at a time when subaqueous drilling was an art little understood.

Cost of Excavation, Eagle Harbor, Mich.—The following facts have been abstracted from a report by Mr. L. Y. Schermerhorn: In 1877 a dredge was used for removing blasted rock in Eagle Harbor; 3,200 cu. yds. being dredged in 63 days (10-hr.) to a depth of 14 ft. The dredge scow was 65 ft. long, the dredge being an "Otis" with a 1 cu. yd. dipper. The rocks handled by the dredge dipper averaged less than 1 cu. ft. in size. Rocks of 1 cu. yd. or more were chained out. The rock was a trap and conglomerate, weighing 169.4 lbs. per cu. ft.; and 1 cu. yd. of solid rock made 1.83 cu. yds. of loose rock in the scows. The rock dipped 30° to the north. Table XXXVII. gives the data of three seasons' work:

TABLE XXXVII.

Season.	No. of holes.	Ft. drilled.	Av. depth of hole.	Av. dist. apart.	Bits sharpened.	Ft. drilled per hour.	Drill steel, lbs.	Holes exploded.	Dynamite. No. 1, lbs.	Dynamite. No. 2, lbs.
1875..	392	2,099	5.35	5.0	656	2.36	51	147	75	441
1876..	309	1,945	6.30	7.7	132	2.67	40	275	775	3,260
1877..	183	1,343	7.34	8.5	88	2.26	32	154	600	1,899
Total.	...	5,387	....	...	876	....	123	...	1,450	5,600

Most of the drilling was done from a large platform; but for drilling boulders a tripod platform was used, on which the drill was mounted. Drill holes were plugged with wooden plugs, but storms and ice caused the loss of nearly two-thirds of the holes drilled the first season (1875). One-half the cost of drilling was chargeable to the first 2 ft. of the hole; that is, up to the point where the drill pipe entered the hole, protecting it from further filling up with sand. During the last season (1877) the holes were drilled 20 ft. below water surface, or about 4½ ft. below the bottom of the intended excavation; but a greater depth would have made the dredging easier by breaking up the rock better. No. 1 dynamite broke the rock up well for a small area around the hole; but No. 2 broke the rock up better for a greater area. A mixed charge of No. 1 and No. 2 proved the most effective. The following are the data of dredging: June 26 to Sept. 6, 63 days; days worked, 47; average hours worked per day, 12¼; repairs made during good weather, 83 hrs.; repairs made during bad weather, 46 hrs.; rock dredged per hr., 5.55 cu. yds.; rock dredged (place measure), 3,000 cu. yds.; boulders dredged, 200 cu. yds.; large rock chained out by divers, 150 cu. yds.; average depth of solid rock excavated, 2.6 ft.; maximum depth of rock, 5.0 ft.; area excavated, 35,000 sq. ft.

Cost of Rock Excavation, Pier 14, New York Harbor.—In the *Trans. Am. Soc. C. E.,* Vol. XXXII., 1894, Mr. John A. Bensel describes the method and cost of excavating 1,530 cu. yds. of mica-schist near Pier 14, New York City. The excavation was carried to 35 ft. below low water, or 40 ft. below high tide. A contractor began the work at $25 per cu. yd., but finally abandoned the work. The crew consisted of 1 diver, 1 foreman, 2 blacksmiths and 5 deck hands; and, drilling from a platform with one Ingersoll drill (largest size), they averaged only 13¼ lin. ft. a shift. Two 18 x 20-ft. platforms, floated out on pontoons and supported by 8 x 8-in. spuds 55 ft. long, were used. When standing on the spuds the platforms shifted up and down stream with the tides, 3½ ft. from the vertical. The platforms collapsed three times; once due to swell of a passing boat; once due to a blast; and the third time without apparent cause. While the platforms were being repaired, a crew of 2 divers and 10 men removed 85 cu. yds. of the blasted rock, at the rate of only 3.4 cu. yds. of loose rock per shift. Then a grab-bucket dredge was tried; and averaged 18 cu. yds. of loose rock a day for a week. The contractor then abandoned the work. The Dock Department then built a large four-drill scow of 12 x 12-in. spruce, the dimensions being 22 x 33½ ft. x 6 ft. deep. The scow and plant cost $5,000. This scow was not provided with spuds (a fatal omission), but was anchored with four pile-driver hammers of 3,000 lbs. each. The drill rods consisted of two pieces of 1½-in. octagon steel, joined by a double ended chuck to make a total length of 45 to 50 ft. From Sept. 2, 1892, to Mar. 13, 1893, drilling was carried on without interruption (presumably working one shift a day with four drills), and only 231 holes (2½ and 3-in.) were drilled, each averaging 7 ft. deep, which was 3 ft. below grade. Gelatin (95 per cent.) was used in blasting. In dredging, clam-shell buckets, 4 and 7 cu. yds. capacity, a grapple and a bucket dredge were tried, with little difference in results, all being very disappointing. The solid rock dredged was 1,530 cu. yds., plus 450 cu. yds. of riprap, and it took 886 hrs. work of the dredge to do this work, costing $22,145 for the dredging alone! The amount of rock removed by the dredge was 4,805 cu. yds., measured in the scow; beside which 480 cu. yds. of rock (scow measure) were removed by divers. The total cost of this work was nearly $70,000. I think it would be hard to find a better example of money wasted in an attempt to do work by day labor instead of by contract. The failure of an inexperienced contractor evidently led the Dock Department into an expensive experiment. Another feature about this work that showed lack of experience was the failure to space the holes closer together, or to drill holes larger in diameter, or both. If that had been done the dredge would have been more effective. The mica-schist of Manhattan Island is an exceedingly tough rock, and it requires close spacing of holes for subaqueous work like this, in order to break the rock into small sizes. The Dock Department, however, spaced the holes 6 ft. apart on the north and south lines, and 5 ft. apart on the east and west lines, according to Mr. Bensel; although according to the scale drawings given by him the distances were 5 ft. and 4 ft. instead of 6 ft. and 5 ft., as stated in the text. The result of the blasting seemed to be to stack the broken stones against each other, and not to loosen and throw up the mass. Divers described this bottom as being oftentimes like a pile of grave stones, one stone lying against another.

Drilling and Dredging Boulders.—The following is an abstract from *Engineering Record,* Jan. 13, 1900: At Wood's Hole, Mass., large boulders were encountered in dredging and were drilled from a platform (10 x 25 ft.) suspended from the dipper boom by a tackle at each corner. Ordinarily the platform is held by clamps which slide on vertical clamp timbers on the bow of the dredge. If it is desired to swing the platform around one end as a center, it is clamped to one guide only, which is in the middle or at one corner. A slot to drill through runs from end to end of the platform.

A 3½-in. pipe has at one end an 18-in. ring 6 ins. deep with the annular space cast full of babbitt; and this is set vertically in the slot of the platform. The loaded end of the pipe is set on the highest point of a boulder, and, even against the strongest tides, is held in position by its weight and by guys from the bottom to the platform. A Rand drill is placed over the pipe and its 2¼-in. bit inserted in it. The Rand drill is lowered by tackle when the limit of its feed is reached. For charging the bit is replaced with a 2-in. loading pipe and a 1-in. cartridge is inserted and fired without moving the dredge. An ordinary day's work with a 7-yd. dipper dredge has been 126 cu. yds. of earth and 7½ cu. yds. of rock.

Drilling in San Francisco Harbor.—The following is an abstract from *Engineering Record,* May 26, 1900: In San Francisco Harbor ledges of metamorphic sandstone were drilled from a revolving platform. The platform, 25 x 160 ft., is made of four lines of longitudinal stringers of 3 x 12-in. timbers bolted together. These stringers rest on 8 x 10-in. floor beams that are 20 ft. c. to c., and queen-post trussed. Through the center of the platform rises a mast 2 ft. square and 68 ft. high, made of four pieces of 12 x 12-in. bolted together and dressed to a diameter of 18 ins. for the upper 15 ft. The top of the mast is guyed by four wire cables to anchors. One block of a tackle is clamped to the upper end of each guy; the other block is attached to the top of the mast, and enables the guys to be tightened or adjusted readily. Fifteen feet below the top of the mast there is fixed to it a collar with a channel in its upper surface, in which there are steel balls for the bearings of a revolving collar above, to which are attached 18 guys (1-in.), or supports, one to each end of each of the nine floor beams. These guys are adjusted by long turn-buckles at the bottom. Two steam drills and one well driller are installed on the platform and operate 10-in. bits. The holes are cased with 10-in. sheet iron pipe. Steam boilers are located on a barge and deliver steam through ball and socket pipe.

CHAPTER XVI.

COST OF RAILWAY TUNNELS.

The American System.—In America there is to-day only one system of rock tunneling in common use. The American system has four distinctive features: (1) An advance tunnel called a "heading" is driven in the upper part of the proposed tunnel. (2) The drill holes in the center of the heading converge in pairs, forming a V, and are fired first so as to form a wedge-shaped cut, called the "center cut." (3) The lower part of the tunnel, called the "bench," is taken out by blasting in holes drilled vertically, or nearly so. (4) Timbering, if used at all to support the roof, consists of "bents," each having upright posts supporting a segmental wooden arch, upon which rests longitudinal planking, called "lagging."

Two of these four features appear unquestionably to have been invented by Americans. The center-cut system of drilling was first used in the celebrated Hoosac Tunnel in Massachusetts (begun 1858); and the segmental arch system of timbering, which leaves the entire area of the tunnel unobstructed by props or supports of any kind, was first used in 1854 in the Van Nest Gap Tunnel, on the D., L. & W. R. R., in New Jersey, under Mr. James Archibald, Chief Engineer. Three rafter pieces had been used in American tunnels prior to 1854, but to Mr. Archibald belongs the credit of having increased the number so as to make a regular wooden arch. This system of timbering has been successfully used even in tunneling through sand and gravel, and it is an open question whether any of the cumbersome systems used abroad need ever be used in America.

Occasionally some contractor tries an experiment such as driving the "heading" at the bottom instead of at the top. After killing a few men, return is usually made to the top heading. It is exceedingly difficult, and in loose rock impossible, to prevent disastrous falls of rock from the roof where the bottom heading method is used. One contractor, an eminent member of the American Society of Civil Engineers, had his life crushed out by a fall of rock in a tunnel that he was driving in New York City by the bottom heading method.

Fig. 51 shows the American system of timbering.

When machine drills were first used in tunneling it was thought necessary to mount the drills on drill carriages which ran upon tracks. The delays involved in clearing away the muck (as the broken rock is called) so that the drill carriage could be run up to the face again after blasting, led to the invention of the column method of mounting drills. Two drills may be mounted on opposite sides of one column, and I have seen two columns with four drills used in a heading 10 ft. wide. Two drills on one column, however, cannot be worked to advantage in a heading much less than 6½ ft. wide.

With four drills at work the holes, which are usually 8 to 12 ft. deep, are drilled in a heading with great rapidity; but the time lost in waiting for smoke to clear away after blasting, and in mucking (loading the rock), usually consumes 50 per cent. of the total time. In railway tunneling rapid progress is usually imperative, and this is especially true where tunnels are so long that the rest of the construction work will be completed ahead of the tunnel work. To overcome the delays incident to blasting and mucking, the contractor should spare no reasonable expense where rapid progress is desired; yet one of the commonest errors is to provide no adequate means for clearing the air after blasting. I have described the Simplon Tunnel work in some detail, because I think that American contractors are not, as a rule, awake to the advantage of using water under pressure to lay the dust and the smoke, in addition to using properly designed fans or blowers. Another lesson that can be learned from the Simplon Tunnel is the use of sheet iron covering laid down upon the car rails before blasting. By doing this it is possible for the muckers to clear the track quickly after a blast, so as to get muck cars up to the face without delay. A third lesson to be learned is the use of shallow holes with very heavy charges of explosive. This causes the muck to be broken up fine and to be hurled away from the face, so that it takes the minimum of time to clear the face ready to begin drilling. I repeat that the contractor who seeks to make a record in rapid tunneling should bend every effort to provide ways and means for the rapid clearing of muck, gas and dust away from the face, so as to keep the drills at work as large a percentage of the time as possible.

A Device for Laying the Dust with Water.—Where a long tunnel is to be driven it will unquestionably pay to have a water main laid alongside the air main, so as to deliver a stream of water against the face after a blast; but in short tunnels it may not pay to do this. A device that has been used with success in England by Mr. William James is described in a paper read May 19, 1904, before the Institution of Mining and Metallurgy. As originally designed it consists of a length of 6-in. pipe, which is let into the 2-in. air main at the mouth of the level (in a mine). This 6-in. pipe is provided with a tap, through which it can be filled with water from a cistern just before blasting. After blasting a compressed air valve is suddenly opened, and the water is carried by the air in the form of a fine spray out of the air pipe, which is directed against the face. The result is that for 40 ft. back of the face the dust and nitrous fumes are quickly laid by the water. A ventilation pipe into which air jets are delivered quickly sucks out the CO and CO_2 gases, thus leaving a face clear of dust and gases.

The Gallitzin Tunnel.—This is a single-track tunnel, 3,600 ft. long, through the Alleghenies, on the line of the Pennsylvania R. R. The work was done in the years 1903 and 1904, under contract, by Mr. P. F. Brendlinger, M. Am. Soc. C. E., who was kind enough to give me every facility for studying it. The material encountered was mostly shale and some sandstone. Figs. 48 and 49 show the tunnel dimensions and the placing of drill holes. For the most part no timbering was required. Two portal headings were

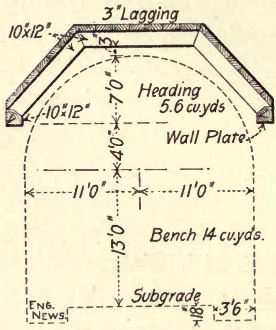


Fig. 48.

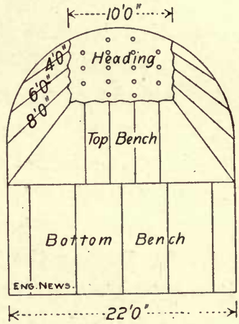


Fig. 49.

driven, there being no shafts. In each heading there were four air drills mounted on two columns. Headings were at first 7 ft. high and 15 ft. wide, but were gradually narrowed, to 10 ft. wide. Each driller was given a day's stint of four 10-ft. holes in the 10-ft. heading, although when the heading was 15 ft. wide five holes was a shift's work per drill. The heading force began work at 7 A. M., but drilling did not usually begin before 8 A. M., and was finished by 3 P. M. The holes were loaded and fired by 4 P. M. The drilling force on each heading consisted of 4 drillers, 4 helpers, 1 nipper (or tool carrier), 1 mucker, 1 timberman, 1 powderman and 1 foreman, a total of 13 men. The night shift on the heading consisted of 14 muckers, 1 foreman, with 2 mules hauling the dump cars to the bench, which was never more than 1,000 ft. back of the face. At the bench the muck was dumped and loaded by a steam shovel, together with the bench muck, into cars hauled away by a dinkey locomotive. Although the holes in the heading face were 10 ft. deep, the actual advance after each blast was 9 ft. The 6 "cut holes" were fired first, then the side holes on one side, then the side holes on the other side, and finally the roof holes, making four separate shots for the 16 holes. Each hole was charged with 14 sticks of 40 per cent. Forcite gelatin, using three boxes for a charge. Each firing threw down about 23 cu. yds., hence there were 7 ft. of drill hole and 6½ lbs. of Forcite per cu. yd. of heading. The muck was broken into small fragments by the blast. Each mucker averaged less than 2 cu. yds. loaded per shift. The average progress was 50 ft. of heading a week at each end of the tunnel, working as described.

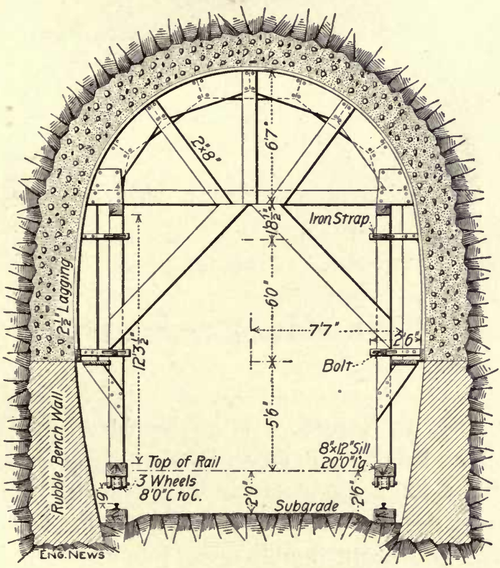


Fig. 50.

The bench was taken out in two lifts, as shown in Fig. 49. Two or three drills on tripods first drill holes to widen out the heading. In the top bench four vertical holes are driven 7 ft. deep, and two diagonal holes, about 7 ft. back of the face. The bottom bench holes are placed 12 to 14 ft. back of the face. The four vertical holes in the top bench are "sprung" by firing 1 to 1½ sticks in each hole, and then loaded with 5 to 8 sticks per hole. The holes in the bottom bench were "sprung" twice; first with two sticks per hole, then with 10 to 14 sticks, and finally loaded with 40 to 60 sticks in each hole. About 1,000 lbs. of Forcite were used per month in both ends of the tunnel. The steam shovel was run with compressed air, and was used in this single-track tunnel with economy. The tunnel was lined with rubble bench walls and a concrete arch, as shown in Fig. 50. Space forbids going into greater detail.

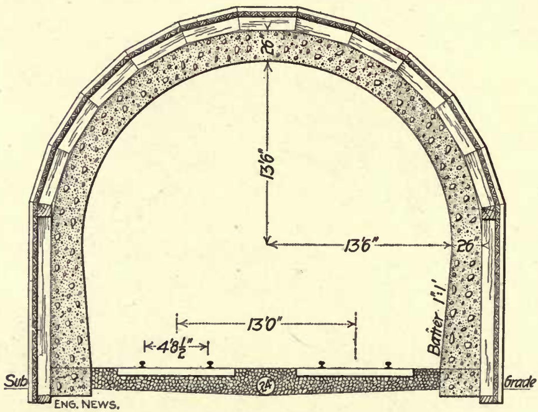


Fig. 51.

Wabash R. R. Tunnels.—I am indebted to Mr. T. H. Loomis, Div. Eng. P., T. & W. R. R. (Wabash system) for much of the following data kindly furnished by him when I went over the line in 1903 studying the methods and cost of excavation. Eight double-track tunnels were underway, the cross-section of each being as shown in Fig. 51. The material encountered was shale, sandstone, fire clay and occasional seams of coal—characteristic of eastern Ohio and western Pennsylvania. The section above the wall plates (*i. e.*, the longitudinal timbers on top of the posts) requires an excavation of 15 cu. yds. per lin. ft. The clear width between wall plates is 34¼ ft. The segmental arch timbers are 12 x 12 ins., lagged with 4-in. plank, the arch ribs being 3 to 4 ft. c. to c. The favorite method of attack, as shown in Fig. 52, was by what I will term the twin-heading method; two 7 x 8 ft. headings being driven as shown, and afterward enlarged. The floor of these headings is 12½ ft. above subgrade, thus leaving a 12½-ft. bench, A C D E, to be taken out. One machine drill is operated in each head-

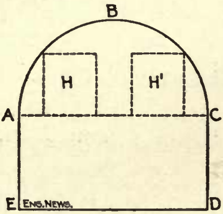


Fig. 52.

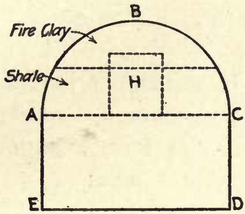


Fig. 53.

ing (two could be worked) for the drilling is easy. The rivalry between the two drilling gangs in these twin headings appeared to me to be one of the best features of this method of attack. It is certain that no hitherto published data show as low a cost per cubic yard for tunnel work as the data which I secured on this work. The weekly progress was not rapid, but, as all the tunnels were comparatively short, there was no necessity of going to great expense in securing rapid progress—a fact that tunnel contractors should bear in mind. Steam drills were used in some of the short tunnels. The following is the actual cost of excavating and timbering the section of a tunnel above the wall plates (15 cu. yds. per lin. ft.), using air drills, for a distance of 100 lin. ft.:

Labor	$2,527.45
2,000 lbs. 40 p. c. dynamite at 12 cts.	260.00
470 gals. kerosene oil, at 12 cts.	56.40
1,875 gals. gasoline, at 12 cts.	225.00
3,000 bu. coal for compressor, at 9 cts.	270.00
Machine and lub. oils	62.5c
Blacksmith shop	150.00

41,649 ft. B. M. timber, at $23	$957.93
Total cost of 100 lin. ft.	$4,509.28
Cost per lin. ft. above wall plates	45.09
Cost per cu. yd., including timber	3.06
Cost per cu. yd., excluding timber	2.60

The material in this case was sandstone.

On another tunnel the section above the wall plates was excavated by hand at a cost $40.90 per lin. ft., or $2.73 per cu. yd., for a distance of 110 ft., the material being hard fire clay in the upper half and shale in the lower half of the section excavated, making easier excavation than in the sandstone. The force engaged in hand drilling, by the twin-heading method, was:

	Wages per 10-hr. shift.
1 general foreman	$4
1 foreman	3
1 blacksmith	3
2 carpenters, at $3	6
10 miners, at $2	20
10 muckers, at $1.50	15
1 team	4
Total per shift (10-hr.)	$55

While these men took out the whole section above the wall plates (15 cu. yds. per lin. ft.) for $2.73 per cu. yd. for labor and explosives (not including cost of timber), working in shale and fire clay, they excavated a 7 x 8-ft. heading in sandstone for $3.75 per cu. yd., distributed as follows:

	Per 10-hr. shift.
Labor on 7 x 8 heading	$18.00
Dynamite	3.84
Repairs	.90
Light	.32
Total per shift	$23.06

Each shift excavated 6.2 cu. yds. of this 7 x 8.-ft. heading, making the cost $3.75 per cu. yd., as above stated, equivalent to an advance of 3 ft. per shift.

No night shifts were being worked on the eight tunnels, and the progress per week in shale was 25 ft. when working by hand and excavating 15 cu. yds. per lin. ft.; and 50 ft. a week working with machine drills. In hard sandstone the weekly progress was about 15 ft. by hand and 30 ft. with machine drills, in all cases working only one 10-hr. shift in the 24-hr. day.

The following is the actual cost of timbering on one job:

	Per M.
Georgia pine f. o. b. cars.	$23.60
Hauling 6 miles	3.00
Cost of framing	5.00
Cost of erecting	3.00
Total per 1,000 ft. B. M.	$34.60

The carpenters received $3 per 10-hr. day, and laborers erecting received $1.50. The cost of framing and erecting, including supervision, was $8 per M., which was about $2 more than it should have cost had there been more workers and fewer bosses. Over the rough roads each team hauled about 1,000 ft. B. M. per load and made one trip of 6 miles each way in a day. The cost of "packing" (*i. e.,* placing small stones) above the lagging was 80 cts. per cu. yd.

We now come to what I have said are the lowest records of tunneling cost yet made public:

Tunnel heading in sandstone, double track full section above the wall plate grade (15 cu. yds. per lin. ft.):

	Cu. yd.
Drilling	$0.60
Explosives	.40
Mucking	.85
Total	$1.85

Tunnel bench in same tunnel:

	Cu. yd.
Drilling	$0.40
Explosives	.20
Mucking	.22
Total	$0.82

The sandstone was very hard, breaking in large blocks, which have to be drilled and shot before mucking. A steam shovel is used in the bench, and material of heading is carried about 400 ft. and dumped over the breast of bench, whence steam shovel loads it along with bench material.

In another tunnel, in a formation of practically level strata of slate, limestone (thin) and fire clay (a stone hard as limestone to drill, but disintegrating in the air) the cost was as follows:

Heading—full double-track sections—all above wall plates:

	Cu. yd.
Drilling	$0.48
Explosives	.30
Mucking	.80
Total	$1.58

Bench—same tunnel—full section:

	Cu. yd.
Drilling	$0.30
Explosives	.20
Mucking	.18
	$0.68

In the case of another tunnel in coal formation with a 5-ft. vein of coal running all through on the wall plate grade; steam drills used in rock, and steam coal augers in the coal, with steam shovel for mucking, the costs were as follows:

Headings—per cubic yard—double track:

Labor	$0.966
Explosives and materials	.090
Total	$1.056

Bench—same tunnel and formation:

	Cu. yd.
Labor	$0.38
Explosives and materials	.04
Total	$0.42

This last may seem too low, but it was in all probability the cheapest material a tunnel is ever built in, and the organization was so good that it was worked with extreme economy. A core of about 2 cu. yds. per lin. ft. was left in the middle of the heading (between the twin headings) and taken out along with the bench.

The Stampede Tunnel.—In *Engineering News,* Oct. 3, 1891, Mr. Charles W. Hobart gives data on the Stampede or Cascade Tunnel of the Northern Pacific R. R. Bids were opened in New York Jan. 21, 1886, for a 2-mile tunnel to be completed in 28 mos. Of the 12 bids, that of Mr. Nelson Bennett was lowest and was accepted. A forfeit of $100,000 and 10 per cent. of the contract price for failure to complete within the time was required. Mr. Bennett telegraphed his general manager to gather men and clear a road to get the machinery on the ground. The plant was purchased for $100,000 in New York and shipped. It consisted of 5 engines, 2 water wheels, 5 air compressors, 5 boilers of 70-H.-P. each, 4 fans, 2 electric arc light plants, 2 miles of 6-in. wrought iron, 2 miles of water pipe, 2 machine shop outfits, 36 air drills, 2 locomotives, 60 dump cars, 2 saw mills and other necessaries. This plant had to be transported on wagons and sleds from Yakima, Wash., a distance of 82 miles to the east portal of the tunnel and 87 miles to the west portal. The first wagon loads started Feb. 1, and the first boiler Feb. 22. By June 19 the plant for the east portal, and by July 15 the plant for the west portal had reached its destination. On Feb. 13 hand drilling was begun on the east portal and 411 ft. of tunnel had been driven when the machines began June 19. On March 15 hand drilling started at the west end and by Sept. 1, when the machines started, 488 ft. had been driven. The last 15 miles of the hauling before reaching the mountains was in mud, so that wagons were hauled by block and tackle, planks being laid down in front of the wheels and taken up as fast as the wagons passed. About one mile a day was covered in this way. When the mountains were reached sleds were improvised and hauled by block and tackle with teams. Wagons lightly loaded with provisions traveled 12 miles a day.

The cost of clearing the way and getting the machinery and materials on the work was $125,000, * and 6 mos. time was required. The tunnel was to be 9,950 ft. long, 16½ x 22 ft. in the clear; 900 ft. had been driven by hand, leaving 9,050 ft. to be driven in 22 mos.

An 8-ft. heading was driven along the top of the tunnel and was kept 30 ft. ahead of the bench. The tunnel was timbered as work progressed. The average number of men employed, after the machinery was installed, was 350. They worked 10-hr. shifts, receiving $2.50 to $5 a day. Contractor boarded men at 75 cts. a day. A bonus of 25 cts. a day was paid each laborer for every foot gained during the month over the necessary average of 13.6 ft. a day in both headings combined, and each driller received a bonus of 50 cts. per day per ft. gained. Every day of the year was worked, requiring two shifts of 75 men each, beside the engineers, firemen, carpenters, machinists, etc., making a monthly payroll of $30,000. The best month's progress was April, 1888, when a total advance of 540 ft. was made in the two headings, or 9 ft. a day per heading. The average progress for 21 $^2/_3$ mos., with power drills, was 413 ft. per

NOTE.—Wages were $2.50 for laborers, which is an unusually high price.

month for the two headings. On May 3, 1888, the headings met, and on May 14 the excavation was completed, 7 days before the time limit. The track was laid in two days more and on May 22 the first regular train passed through the tunnel.

The total explosives used were 309,625 lbs., as follows:

	No. of 50-lb. boxes.
Giant No. 1, 60 per cent.	403¾
Giant No. 2, 45 per cent.	2,123½
Hercules No. 1, 60 per cent.	1,609½
Hercules No. 2, 45 per cent.	1,781¾
Nitro glycerin No. 2	232
Forcite No. 2	41½
Total No. of 50-lb. boxes	6,192

The average price of all explosives was $10 a box, or 20 cts. per lb. The total number of men killed in the two years was 13. The following data were furnished by Mr. Andrew Gibson, Asst. Engr. The American center-cut system of blasting was used; 20 to 23 holes, 12 ft. deep, being drilled in the heading, and about 18 holes in the bench. Each drill, in medium hard rock, would make 6 or 7 holes in 5 hrs., although at times in an exceedingly hard layer 15 hrs. would be required. About 400 lbs. of dynamite were used at each blast in each of the headings and benches. This would break 8 to 12 lin. ft. of tunnel, although in very hard rock at times only half this progress was made. The rock is basaltic, * with a dip of 5° to the west. It required immediate timbering, which delayed the drillers and muckers about 25 per cent. of the time. During the period of hand drilling there were 17 men, with about 23 muckers, employed in each heading, and 4 lin. ft. of tunnel in 24 hrs. were averaged. During the period of air drilling, 10 drills were used, 5 in each end, and the progress was 6.9 ft. in 24 hrs. per heading, or 207 ft. per mo. of 30 days. While the contract size of the tunnel was 16½ ft. wide, and 22 ft. from subgrade to face

* Elsewhere it is stated that the rock was shale.

of arch, the timbered sections had to be excavated 19½ ft. wide by 24 ft. high, thus requiring 15.7 cu. yds. of excavation per lin. ft. where timbering was used, as against 12.36 cu. yds. where no timber was used. Timbers were 12 x 12 ins., except the 8 x 12-in. sills. Five segments were used in the arch, lagged with 4 x 6-in. pieces. Bents were spaced 2 to 4 ft. Water gave no trouble.

Mules were used for hauling up to the first half mile; then small locomotives, which hauled 8 to 12 cars. A "go-devil" or platform on wheels was used to great advantage in loading cars. The men wheeled the rock on plank runways from the heading to the "go-devil," dumping directly into cars below; and the muckers on the heading never interfered with those on the bench. It was also a great convenience in timbering. Before blasting the drills were loaded upon the "go-devil," and it was pushed back some distance from the face. Endless belt conveyors for removing muck to the "go-devil" were contemplated, but they were never used, as with the large force of men at work they would have been in the way.

The swelling of the shale on exposure often reduced a 12-in. timber to 4 ins.; hence it was necessary to line the tunnel with masonry. Concrete side walls and a brick arch were used for lining. The concrete mortar was brought in on cars and run back of the forms through spouts, without shoveling; then the broken rock was shoveled into the mortar from a flat car.

The total cost of the tunnel to the N. P. R. R. under Mr. Bennett's contract (which did not include masonry lining) was $118 per lin. ft. Mr. Bennett's brother was the superintendent of the work. The actual cost of tunneling the west end during the month of Nov., 1887, was $75.75 per ft. for the 258 ft. driven, distributed as follows:

Labor.	
Supt., ½ mo., at $500	$250.00
" 1 mo., at $250	250.00
Master mechanic, ½ mo., at $150	75.00

Engineers,	4 x 30 = 120	days	at	$4	....	$480.00
Machine repairers,	3 x 30 = 90	days	at	$3.50	....	315.00
Firemen	4 x 30 = 120	"	"	2.50	....	300.00
Blacksmiths	2 x 30 = 60	"	"	4.00	....	240.00
" helpers	2 x 30 = 60	"	"	2.50	....	150.00
Carpenters	396	days	at	$3.00	....	1,188.00
Foremen	160	"	"	4.50	....	720.00
Drillmen	294	"	"	3.50	....	1,029.00
Chuckmen	293	"	"	3.00	....	579.00
Muckers	1,138	"	"	2.75	....	3,129.50
Nippers	60	"	"	2.50	....	150.00
Dumpmen	60	"	"	2.50	....	150.00
Car drivers	60	"	"	2.50	....	150.00
Time keeper	30	"	"	2.50	....	75.00
Lampmen	60	"	"	2.50	....	150.00
Laborers	662	"	"	2.50	....	1,655.00
Bonus for daily progress over 6 ft.					....	800.00
Total labor for 258 ft. at $45.90 per ft.					.....	$11,835.50

Material.

78,000 ft. B. M. timber, at $10	$780.00
800 lbs. wrt. iron, at 6 cts.	48.00
64½ cords wood, at $3	193.50
240 tons coal, at $4	960.00
900 caps, at 1 ct.	9.00
14,400 ft. fuse, at 1 ct.	144.00
13,800 lbs. dynamite, at 16 cts.	2,208.00
Total materials for 258 ft., at $16.80 per ft.	$4,342.50

Plant.

6 per cent. of $50,000 plant, 1 mo.	$250.00
1/28 of 75 p. c. depreciation * of $50,000 plant......	1,339.28
10 p. c. on all above to cover all possible omissions.	†1,776.72
Total plant charges for 258 ft. at $13.05	$3,366.00

* Note that a liberal but not unusual allowance is made for plant depreciation.
† This 10 per cent. practically covers the cost of installing the plant.

Summary of cost per ft.

Labor	$45.90
Material	16.80
Plant	13.05
Total	$75.75

During this month the entire length was lined with timber, the rock being a soft basaltic rock that drills well but goes to pieces rapidly on exposure. There were no accidents or delays.

On the east end during this same month, with an equal force, the progress was 246 ft., at a cost of $72.70 per ft. It will be noted that wages were high. It will also be noted that the cost of hauling and installing the plant is not included, although a liberal allowance is made for plant depreciation and in the 10 per cent. added to cover omissions.

The contractor received for his month's work on the west end of the tunnel:

258-ft. tunnel, standard section, at $78	$20,124
862 cu. yds. extra excav., at $4.50	3,879
78,000 ft. B. M. lining, at $35	2,730
258 ft. of tunnel, timbered, at $103.62	$26,733

The best month's record in driving a heading was 274 ft., but, as before stated, the average progress with the air drills was 207 ft. per mo. per heading, although in the month of Nov., 1887, 258 ft. were progressed on the west end, which was 25 per cent. better than the average progress. Assuming that 15.7 cu. yds. were excavated per lin. ft. of tunnel, the total excavation at the west end for November was 4,052 cu. yds. It is probable that the 862 cu. yds. extra excavation, above given, are included in this estimate, because the "standard section" differed from the timbered section by 3.3 cu. yds. per lin. ft., and in 258 ft. this would amount to 852 cu. yds. On this assumption (of 4,052 cu. yds.) the labor cost $2.92 per cu. yd.; the materials, $1.07 per cu. yd.; and the plant, \$0.83 per cu. yd.; total, \$4.82 per cu. yd. for the best month's work.

Mount Wood and Top Mill Tunnels.—Mr. W. J. Yoder, in the Journal of the Western Society of Engineers, 1897, gives the following data: The tunnels (built in 1888-1889) are within the northern city limits of Wheeling, W. Va., and the material penetrated was for the most part shale of the coal measures. The shale disintegrates rapidly upon exposure and must be supported. The block or American system of timbering was used for lining, and was kept never more than 50 ft. back of the face. All drilling was done by hand, and the position of drill holes is shown by sketches in the article. A top heading 10 x 34 ft. was driven, and then widened; the bench was taken out in two lifts. The first or cut holes in the heading were drilled so as to blast out a long horizontal wedge of rock near the roof; these holes being 5 to 6 ft. deep. Then a lower row of 5-ft. lift holes was fired. Finally the bottom of the heading was taken out like a bench by a row of vertical holes and a row of horizontal holes. In all 33 holes were fired in the heading, aggregating 160 lin. ft., and requiring 60 lbs. of 40 per cent. Forcite to load them. The effect of the firing was to make an advance of 2½ ft., displacing 25 cu. yds. The heading gang consisted of 1 foreman, 14 drillers, 12 muckers and 1 nipper. About 25 lin. ft. of drilling was considered a day's (10 hrs.) work for 2 men. The muck was wheeled in iron barrows to a traveler and dumped down chutes into cars. The heading gang timbered and placed the packing above the arch; two 10-hr. shifts per week being needed for this work, leaving 10 shifts per week for advancing the heading. The timbering is fully described; 660 ft. B. M. of white oak were used per lin. ft. of tunnel. The bench holes were 8 ft. deep, churn drills being used except for the corner holes and for blockholing. The bench force consisted of 1 foreman, 6 drillers, 18 muckers, 2 mule drivers, 3 dump men and 1 nipper. The average haul was about 800 ft.

The maximum monthly progress (working two 10-hr. shifts) in a heading on the Mount Wood Tunnel was 130 lin. ft., the average monthly progress being 84 ft. The maximum monthly progress on the bench was 125½ ft., the average being 97 ft. The average excavation was 10.2 cu. yds. per lin. ft. of heading and enlargement, and 18 cu. yds. per lin. ft. of bench. The total excavation in both tunnels was 49,670 cu. yds., and the excavation in approaches was 25,751 cu. yds.

The number of men employed was 350. The heading men were composed of two-thirds Negroes and one-third Austrians. The foremen were Irish. The best drillers were Negroes. No work was done Sundays or Saturday nights. The scale of wages (10-hr. shift) was as follows:

Heading Gang.

1 foreman	at	$4.00
14 drillers	"	1.75
10 muckers	"	1.50
1 nipper	"	1.25

Bench Gang.

1 foreman	at	$3.00
6 drillers	"	1.75
16 muckers	"	1.50
2 men (lagging)	"	1.50
1 nipper	"	1.25
2 drivers	"	1.50
3 dumpmen	"	1.50
2 mules		

Miscellaneous.

1 carpenter	at	$2.50
4 sawyers	"	1.75
1 trackman	"	2.50
3 blacksmiths	"	3.00
1 walking boss	"	4.00
1 timekeeper	"	2.25
1 engineer and fireman	"	2.50
1 electrician	"	2.50

Cost of Labor per Lin. Ft. of Tunnel.

Labor excavating (heading, $22.79; bench, $20.95)	$43.74
Hauling and dumping	5.65
Labor timbering	4.19
" framing timber	.77
Blacksmithing	1.00
Track repairs	.21
Labor electric lighting	.88
Superintendance and accounts	2.00
Total labor	$58.44
Cost per cu. yd.	2.06

The above does not include the cost of timber, oil, fuel, wear of tools or explosives. About 1 lb. of 40 per cent. Forcite was used per cu. yd. of tunnel excavation, or 28 lbs. per lin. ft. The labor cost was $2.34 per cu. yd. of heading, and $1.10 per cu. yd. of bench excavation, making an average of $1.55 per cu. yd., not including the items of timbering, etc. The labor cost of erecting arch and packing back of it was $3.19 per lin. ft. of tunnel; or $7.80 per 1,000 ft. B. M. The labor cost of erecting plumb posts and side lagging and packing same was $2.33 per lin. ft.; or $4.27 per 1,000 ft. B. M. The contractors were Paige, Carey & Co., of New York, whose superintendent was Mr. Frank Moran.

Tunnel Driven by Hand on the B. & O.—In *Engineering News,* April 5, 1894, Mr. J. G. G. Kerry gives description and cost of a short tunnel built in 1891 on the W. Va. & P. R. R., a feeder of the B. & O. system. The tunnel is on a ¼ per cent. grade falling to the south, with a length of 624 ft., in a soft blue clay shale, nearly dry and showing little stratification. This shale disintegrates rapidly on exposure. The width was 23 ft., height from floor to spring line 13 ft.; semi-circular arch of 11½ ft. radius. The area of the heading was 208 sq. ft.; bench, 299 sq. ft.; total, 507 sq. ft. Work was all done by hand. The heading gang consisted of 1 foreman, 8 miners, 6 muckers and 1 nipper. Common labor ers were paid $1.45 and miners $1.75 per 10-hr. day. Three sets of holes (2 wet and 1 dry) were drilled in the heading; each set consisting of 4 holes about 4 ft. deep; and 24 ft. of hole was considered a good day's work for two miners. Each hole was loaded with 4 to 6 sticks ($^1/_3$ lb. per stick) of dynamite; and the average advance from a blast was 2½ ft. A scaffold car, or go-devil, was used in handling the muck. It was provided with a derrick and also used for handling timbers, lagging and packing.

The bench gang consisted of 1 foreman, 8 drillers, 10 muckers and 1 nipper. The bench was shot down in 4-ft. holds or lifts, two half-depth blasts being made for each hold. Each blast consisted of four holes, two being center holes, and two nearly vertical under the wall plate. The charge was 10 sticks to an outside hole and 15 sticks to a center hole. Muck was taken out in 1 cu. yd. dump cars in trains of two. Stone flat cars with platforms flush with top of wheels were used for handling large rocks. The bench was kept two wall plate lengths behind the heading, making the same progress, 2½ ft. per shift. The actual excavation was at the rate of 5 ft. per shift, but the time consumed in pointing down projections, timbering and packing being equal to the time spent in excavation, reduced the average progress to 2½ ft. per shift. The work was done by contract, and it cost the company at contract prices as follows:

11,726 cu. yds. of excavation	at $2.85	$33,419
742 " " " packing	" 1.75	1,298
256 " " " fallen rock	" 1.25	320
303,000 ft. B. M.	" 30.00	9,090
Total 624 lin. ft. of tunnel	at $70.70	$44,127

The actual cost to the contractor was about $35,000.

The method of handling and placing the segmental arch timbering is described in detail. The timbering consisted of a 7-segment arch of 12 x 12-in. white oak resting on 12 x 14-in. wall plates on top of the posts. The 16-ft. wall plates were jointed by halving for a foot at each end, so that the forward end always showed the lower half of the joint. The arches were 8 ft. c. to c. The segments of the arches were erected on temporary centers made of 2-in. plank. These centers were erected in two parts and joined at the crown by bolts; a long dog-hook, fastened to the center, was driven into the preceding arch to hold it in place laterally. The arch timbers were wedged solidly against the roof, and the centers withdrawn. The lagging was close laid, all voids being packed with broken sandstone.

Each end of the tunnel was lined with masonry for 50 ft., the centers used in this lining being 25 ft. long and mounted on rollers. During use the centers were supported on wedges, which upon being struck lowered the center enough to clear the rock-faced voussoirs. A hole was left in the crown of the arch-center lagging so that the voussoirs could pass through. Above this a piece or two of the tunnel lagging was removed, and an iron bar placed on the timber arches. A set of blocks was hung from this iron bar, and used to raise the voussoir stone. Gas pipe rollers were put under the stone to roll it to place on the center lagging. The stone was then canted up, and a rope slung around it, six men then sliding it to place.

The contract prices were $9 per cu. yd. for portal masonry, $8 for side walls and $14 for arch sheeting. The cost at contract prices per lin. ft. of that part of the tunnel which was lined (excluding portals, fallen material, etc.) was:

Excavation	$53.55
Packing	2.08
Timbering	14.75
Side walls	20.56
Arch	21.42
Packing	2.08
Total per lin. ft.	$114.44.

New Croton Tunnel.—In *Trans. Am. Inst. Min. Eng.*, Sept., 1890, Mr. J. P. Carson gives the following data: Work was begun in 1885 on the New Croton Aqueduct Tunnel, which was nearly as large as a single track railway tunnel. The following were the areas of the excavation:

	As shown on plans.	Actual excavation.
Area above the spring line	73 sq. ft.	105 sq. ft.
Area below " " "	131 " "	178 " "
Total area	204 " "	283 " "

The tunnel was lined with masonry so as to have an inside width at the spring line of 14 ft. and a height of 14 ft. The contract price was $7 per cu. yd. of excavation, or $52.88 per lin. ft. A top heading 8 x 16 ft. to 9 x 17 ft. was run by the American center cut system of drilling. Several shafts were sunk and the tunnel driven from them. Sinking shaft 13A a distance of 110 ft. in wet, soft material cost $416 per ft. Cars holding 1 cu. yd. of broken stone, or ½ cu. yd. of solid stone, were hauled in trains of 2 or 3 cars by one mule. Experiments were tried to determine the relative advantage of driving the heading and bench separately as compared with driving them as is usual, at the same time, and about 60 ft. apart. It was found that in mica-schist when the heading and bench were driven together the average progress of completed tunnel excavation was 30 ft. a week from one face. The average weekly progress of completed tunnel excavation was only 20.4 ft. per week when the heading was first driven through to the next shaft, and then followed by the bench, which was driven through separately. Out of 76,490 ft. of tunneling 23 per cent. was driven by working the heading and the bench separately, causing a very material delay in completing the tunnel.

Average Weekly Progress in Division 3.

	Total No. of Weeks.	Lin. Ft. per Week.	Area. Sq. Ft.
Mica-schist*	427	30.0*	283

* Heading and bench driven at the same time.

Limestone*	95½	22.4*	283
Decomposed rock*	152	9.8*	361
Sand and boulders*	62 1/3	4.25*	490
Soft ground, clay, sand, mica*	128	1.40*	508
Mica-schist†		47.0†	165
Limestone†		33.8†	105
Decomposed rock†		23.5†	105
Mica-schist‡		36.0‡	178
Limestone‡		27.0‡	178
Decomposed rock‡		12.1‡	196

It is interesting to note that although faster heading progress was secured by driving the heading clear through before beginning the bench, the progress was not enough faster to make up for the delay in waiting to begin the bench; so that in mica-schist the average progress per week of heading and bench was $(47 + 36) \div 2 = 41\frac{1}{2}$ ft., and as they were worked separately the final average of completed tunnel was $41\frac{1}{2} \div 2 = 20.7$ ft. per week.

The men decided to break the record for a week's run in the South heading from Shaft 15, beginning at a point 3,200 ft. from foot of shaft. The heading was 9 x 17 ft. in gneiss. Plant: 1 duplex Rand compressor, class "B," rated at 1,325 cu. ft. per min., at 80 lbs. pressure; 4 and 5-in. air pipe; loss of pressure by friction in 3,400 ft. of pipe was 15 lbs.; 3 slugger drills; one 10-hr. shift per day:

7 to 9:30 A. M.	Mucking	2½ hrs.
9:30 A. M. to 4:30 P. M.	Drilling	6 "
4:30 to 5	Charging	½ "
5 to 6	Firing	1 "
Total		10 "

* Heading and bench driven at the same time.
† Heading driven alone.
‡ Bench driven alone after completing heading.

Distance of heading run in a week, 102 ft. (best previous, 90 ft.); area, 145.5 sq. ft.; total, 550 cu. yds.; 25 per cent. of the muck left in tunnel

8 center-cut holes, 10 ft. each	80 ft.
12 side " " 8 " "	96 "
Total ft. drilled per shift	176

Total ft. drilled 2,288, or 4.16 ft. of hole per cu. yd. Total powder, 2,200 lbs., 4 lbs. per cu. yd., or 8.46 lbs. per hole; 13 blasts fired with average advance of 7.86 ft. Crew: 1 heading boss at $3.25; 3 drillers at $2.50; 3 helpers at $1.75; 1 nipper at $1; 1 powder man at $1; 1 muck boss at $2.50; 9 muckers at $1.50; 3 drivers and mules at $2.25; 1 trackman at $1.50; 2 sumpmen at 75 cts. Outside labor: 2 bellmen at 75 cts.; 2 topmen at 75 cts. Power, etc.: 1 engineer at $15 a week; 1 engine driver, $9; 1 fireman, $12.25; 1 electrician, half week, $6; 1 machinist (half week), $7; 1 carpenter (half week), $7.50; 1 blacksmith, $15; 1 blacksmith helper, $10; 1 time keeper, $14; 1 general foreman, $35.

Record Run South Heading.

	Cu. yd.
Drill crew	$0.42
Muck crew	.50
Transporting	.31
Outside labor	.09
Enginemen, etc.	.18
General foreman	.06
Total labor	$1.56
5 tons coal at $5 (7 days), $175;	
Oil and candles, $12.50; steel, $5	.35
2,200 lbs. rackarock at 16½ cts.	.65
Total	$2.56
Interest on plant	.37*
	$2.93

* Too low.

The record week's run on North Heading, Shaft 16 (heading 9 x 16 ft.), with plant same as above, but better overhauled, was as follows: Work was pushed to the utmost for 7 days in mica-schist; 10-hr. shifts. In 10 previous weeks the average had been 46.9 ft. per week. This week's record was 127 ft.; area, 125.6 sq. ft.; 591 cu. yds.; 15 to 20 per cent. of muck left in tunnel. Distance from compressor, 3,965 ft.; loss of pressure, 20 lbs.; 2 Rattler drills.

8 center-cut holes, 8½ ft. each	68 ft.
10 side " " 8 " "	80 "
Total	148 "

Total: 18 shots; 324 holes; 2,664 ft. drilled; or 4.33 ft. per cu. yd.; 2,050 lbs. 60 per cent. powder; or 3.46 lbs. per cu. yd.; or 6.33 lbs. per hole; advance, 7.05 ft. per shot.

Mucking	2.3 hrs.
Drilling	5.5 "
Charging	0.5 "
Firing	1.1 "
Clearing of smoke	0.6 "
Total	10.0

Other Examples of Rapid Heading Work.

Days.	Lin. ft. per day (10 hrs.)	No. of Drills.	Size of Heading
22	10.52	4	14 x 14
26	10.21	4	"
24½	11.75	2	8 x 16
54	10.05	3	"
60	12.16	3	"
7	15.07*	3	9 x 17
60	7.81		9 x 16
7	18.14*	2	9 x 16

Rapid bench work: In 84 days, with 2 drills, averaged 6.85 lin. ft. per day; 3,360 cu. yds.; 4 holes averaging 9 ft.

* Above given in detail.

deep per blast; 4,788 ft. drilled; 1.4 ft. per cu. yd.; 7,500 lbs. powder, or 2.23 per cu. yd.; bench worked at both ends.

The above are record costs. The following are average costs, and are double the costs on the record runs:

Heading, area 105 sq. ft.	Cu. yd.
Inside labor	$3.34
Outside labor	.94
Coal (3 tons day)	.67
Powder at 16½	.50
Interest on plant	.37*
	$5.82
By driving two headings at one time, outside labor cut in two saving	.50
	$5.32

Average cost of bench excavation: Area, 167.4 sq. ft.; 577 lin. ft. in 48 days (two 10-hr. shifts per day); 3,578 cu. yds.; 1 drill boss, $70 per mo.; 1 muck boss, $70 per mo.; 1 drill runner, $2.50; 1 drill helper, $2; 15 muckers at $1.50; 1 driver and mule, $2.25; 1 trackman, $1.50; 2 dumpmen at $1.50; 1 bellman, $1.75; 1 sumpman, $1.75; 1 topman, $1.50; outside labor as above given.

	Cu. yd.
Drilling and blasting (532 holes, 9 ft.)	$0.28
Mucking (3,360 cu. yds.)	.75
Transporting	.20
Outside labor at shaft	.17
Extra " " "	.05
Enginemen, etc.	.17
Machinist, blacksmith and helper	.09
Coal (3 tons. at $5; oil and candles, at $2 per day)	.23
Powder (7,300 lbs. J. L. Aqueduct at 16½ cts.), etc.	.38

* Too low an estimate.

Interest on plant	$0.37*
Total cost per cu. yd. of bench	$2.69

Cost of driving, heading and bench together: 30 lin. ft.; area, 283 sq. ft.; 314 cu. yds.:

	Cu. yd.
Inside labor on heading 120 c. y. at $3.34 } " " " bench 194 c. y. at $1.23 }	...$2.03
Outside labor	.78
Coal, etc.	.62
Powder (3 lbs. per c. y. of heading; 2.3 lbs. per c. y. of bench, at 16½ cts.)	.44
Total for single heading and bench	$3.87†

The Cascade Tunnel.—This tunnel is described in *Engineering News,* Jan. 10, 1901, by Mr. John F. Stevens, Chief Engineer, Great Northern Railway. The tunnel is 13,813 ft. long through the Cascade Mts. on the line of the Great Northern Ry. The width in the clear is 16 ft., and the height from top of rail to bottom of arch is 21½ ft. It was begun, from two headings, Aug. 20, 1897, and completed Oct. 13, 1900. A top heading, 10 x 20 ft., was driven from each end; and the bench was taken out in two lifts. The average monthly progress was 175 ft. at each heading, or 5.76 ft. per day of 24 hours. The best year's work was from June 1, 1899, to June 1, 1900, in which time 5,575 ft. were driven from the two headings, the monthly average being 232 ft. per heading. The best month's progress was 527 ft. from two headings; the best week's progress was 143 ft. from two headings; the best month's progress from a single heading (East) was 301 ft. The rock was medium hard granite, very seamy and very wet. Although hard to drill and blast, the granite disintegrated so rapidly that a temporary timber lining was necessary throughout, and it was afterward replaced with concrete.

* Too low.
† Deduct $0.70 reduced fixed charges when two headings are run.

The work was all done by day labor, no contracts being let. Three 8-hr. shifts were worked. There were 600 to 800 men employed, and they were not very efficient.

Four columns in a heading carried 6 drills (3¼-in. size). From 24 to 28 holes were drilled 12 ft. in the heading, and fired in three rounds by electricity. Including the bench work there were 14 drills used at each end of the tunnel. Rock from the heading and top bench was wheeled in barrows out onto the "jumbo," or "go devil," and dumped through into cars below. A compressed air hoist on the "jumbo" served to lift large rock and to shift the "jumbo" back before firing. Eight electric motor cars were used to haul the muck, etc. One motor hauled 16 to 20 dump cars of 1 cu. yd. each up the 1.7 per cent. grade to the east portal, at 10 miles an hour. The rails were 50-lb. rails laid to a gage of 2 ft.

Large power houses were built at each portal. The east power house contained one Ingersoll-Sergeant duplex compressor, 18 x 24 ins.; one straight line compressor, 18 x 24 ins.; one Rand duplex compressor, 20 x 36 ins.; one Buckeye high-speed engine, 12 x 16 ins.; one Chandler & Taylor high-speed engine, 13 x 14 ins.; six 150-H.-P. boilers; pumps, dynamos, fans and water heaters. Compressed air was delivered through 6-in. mains to the drills, at an initial pressure of 100 lbs.

The tunnel was lined with concrete from end to end, the temporary timber lining being removed. The concrete is nowhere less than 2 ft., and in places it is 3½ ft. thick; spawls and broken stone were packed above the concrete where necessary. To place the concrete without interfering with the muck trains, a platform 500 ft. long was erected, and the cars loaded with concrete were hauled up an incline by a compressed air hoist. The concrete was dumped on the platform and shoveled into the forms. While this was going on another 500-ft. platform was being built in advance. Side walls were built in alternate sections 8 to 12 ft.

long, the weight of the timber arches being thus transferred to the walls. The concrete arch centers were made in 12-ft. lengths, of which there were ten in each end of the tunnel. When the concrete had set the 12-ft. arch center was lowered with screw jacks onto "dollies," pushed forward 12 ft. and jacked up again. Concrete was mixed, 1 cement, 3 sand and 5 parts rock. About 95,000 barrels of Portland cement were used in lining the tunnel, an average of 7 bbls. per lin. ft. of tunnel. Work of lining was begun in Dec., 1899, and finished Nov., 1900; more than 1,000 ft. of lining having been placed in Oct., 1900, in the west end, although the general average was about 600 ft. of lining per month from each end. The tunnel was opened for operation Dec. 20, 1900.

Mr. Willard Beahan, in a letter to *Engineering News*, Feb. 28, 1901, adds some interesting facts about the part of the tunnel driven through soft ground, at the start, where only 0.8 ft. per day was averaged. He also says that it was a serious mistake to have driven the heading in rock by hand 300 ft. in advance of the bench while waiting for the power plant to arrive, for the long heading overtaxed the transportation so that work on the heading had to be stopped until the bench was brought up. The use of four drill columns he regards as novel, and adds that there was plenty of room in which to work six drills, and that it was not necessary to shift any of the columns in drilling a set of holes.

The Kellogg Tunnel.—Mr. U. B. Hough, in *Engineering News*, April 25, 1901, describes a novel method of mucking used in running a large tunnel in the Bunker Hill and Sullivan mines, at Kellogg, Idaho. The entire length of the tunnel will be 9,000 ft., of which 8,400 ft. had been driven at the time of writing. The rock is quartzite and after blasting is loaded by a No. 1 drag-scraper, which is hauled up an incline by a hoisting engine, and dumped through a door into the car. Two scraper loads usually fill a 1-cu.-yd. car, and never more than three loads. After the first car is loaded, the second car is pulled forward under the door.

Five men constitute the mucking gang, one operating the hoist, one attending the cars and three loading and attending the scraper. The time required to remove 40 to 50 cu. yds. of waste is 2 to 2½ hrs. (not stated whether the yardage is solid or loose measure). The incline, or "mucker," is moved by jacking it up (on four screw jacks) off the floor until the weight is on the moving car; it is pushed back until advance rails are laid, then it is pushed forward and lowered to the floor again. The hoist is moved in a similar manner by running a moving car under it. Five men consume only 20 mins. in moving both hoist and mucker. The mucker has been in use 15 mos., and is in fair condition. Two drills mounted on one bar are worked in the heading, which is kept 10 to 20 ft. in advance of the bench. A four-ton electric motor hauls a train of 15 loaded cars. Smoke is removed through a 22-in. pipe (No. 18 iron) by a No. 9 Sturtevant exhaust fan. This fan will remove all smoke in 15 or 20 mins. through 8,000 ft. of pipe. The longitudinal pipe seams are all riveted and soldered; and the joints are wrapped with sheeting and painted with tar. In the month of October, 1898, an advance of 354 ft. was made, but the general average has been 7.5 ft. per day, working three 8-hr. shifts. One-third of the tunnel has been timbered.

The Pryor Gap Tunnel.—Mr. F. T. Darrow describes this tunnel in *Engineering News,* July 3, 1902. The tunnel is single track, on the line of the "Burlington" in Montana, 38 miles south of Billings. It is 500 ft. long, on a curve, through solid dolomitic limestone. The tunnel section outside of the timbering runs 18 cu. yds. per lin. ft., and was taken out by a top heading and two benches, each about 9 ft. high. The work was done by contract, actual tunnel work beginning December, 1900. A 16-H.-P. upright boiler furnished steam to two 3⅛-in. drills in the heading and one drill on the bench. The steam was carried in 2-in. and 1½-in. wrought pipe covered with jointed jacketing; and at a distance of 400 ft. from the boiler no difficulty was found in running the drills. Two 10-hr. shifts were worked daily, or 13 shifts a week; and, when running well, the advance was 50 ft. a week from one heading. The heading gang consisted of 1 foreman, 2 drillers, 2 helpers, 6 or 7 muckers, 1 powderman, beside the fireman and teamster outside. The bench force was about the same, excepting that there was only one driller and helper. In heading work each machine was mounted on a separate column; and a set of 14 holes 6 ft. deep was put in and fired in four rounds, using 60 per cent. dynamite. The south heading was run in 300 ft., and then stopped, and the bench brought up, the heading gang moving to the bench at the north end, which up to that time had been run without bench excavation. The work was completed May 15. Forty pounds of 60 per cent. Giant and 10 lbs. of black powder were used per lin. ft., the black powder being used only in the bench. Timbering was put in after the completion of the excavation while the tunnel was in operation. The particular feature of this work is the simplicity of the plant and the effectiveness of the small gang of men.

The Busk Tunnel.—The following data are given in *Engineering News,* Sept. 27, 1894: The Busk Tunnel Ry. Co. built a tunnel 9,395 ft. long on the Colorado Midland R. R. through the Rocky Mts., 11.7 miles S. W. of Leadville. The contract was let to Keefe & Co., and work was begun Sept. 15, 1890. After all but 921 ft. had been driven the work was turned over to the railway company and finished under the direction of their chief engineer, Mr. B. H. Bryant. The tunnel is single track, 15 x 21 ft., with 10.2 cu. yds. per lin. ft. excavation in rock and 13.8 cu. yds. where timbered. The heading was 7 ft. high and the full width of the tunnel. The first 8 holes, 8 ft. deep, were drilled in two rows from top to bottom, holes being about 2 ft. apart at surface and converging toward the center. The firing of these holes made a V-shaped opening. A second set of holes was drilled parallel to the sides of the tunnel, and when fired the remaining rock was blown into the V-shaped opening. The bench was excavated in the same way. The progress was as follows:

Driving the 2 headings	1,118 days.
Av. daily progress	8.4 ft.
Av. daily progress, best month	10.9 "
Best month's (28 days) progress, one heading	202.5 "

The rock was granite, and in places it disintegrated on exposure, requiring timbering; in other places it was so full of seams as to require timbering; so that 78 per cent. of the tunnel was timbered. The contractor was paid for the tunnel as follows:

9,393 2/3 ft. of tunnel at $62.50	$587,103.75
32,575 cu. yds. enlargement for timbering at $2.50	81,437.50
Cost of timber, 2,723,000 ft. B. M. at $30	81,690.00
Labor timbering at $12 per M	32,676.00
Total 9,393 2/3 ft. at $82.30	$782,907.50

The plant at the Invanhoe end consisted of three 100-H.-P. boilers, two 20 x 24-in. Ingersoll compressors, one 20 x 24-in. Norwalk compressor, one 10-H.-P. engine to drive electric light dynamo, one 20-H.-P. engine to drive a No. 6 Blake blower, 14-in. air pipe, two pumps with 14-in. steam cylinders and 10-in. stroke, six 3⅛-in. Ingersoll drills (4 in the heading and 2 on the bench), a small traction engine running on a 20-in. gauge track hauling nine 3-yd. dump cars. Coke was used as fuel for the traction engine, so that the smoke did not inconvenience the tunnel workmen.

Sutro Tunnel.—Drinker gives in considerable detail the work done in driving the heading of this drainage tunnel in Nevada. The work was begun in 1869 by hand, and in 1874 Burleigh drills were introduced. The heading was 8 x 10 ft., with cut holes 7½ ft. deep and 4½ ft. apart. It is stated that 6 Burleigh drills were operated, but if so they must have been unusually crowded. There were 12 men operating these drills on each shift, and these same men were required to do their own mucking besides. Subsequently 4 Ingersoll drills replaced the 6 Burleighs. The rock was a hard trachite and greenstone. In 1875, working three 8-hr. shifts, the average progress of the heading was 72 ft. a week, or 3.4 ft. per 8-hr. shift; and the best month's advance was 350 ft. During the last 8 mos. of the year 1875 the advance was 2,561 ft., which required 9,882 holes (2 to 2½-in.) averaging 6.77 ft. per hole, making a total of 66,951 ft. of hole to excavate about 7,680 cu. yds., or 8.7 ft. of hole per cu. yd.; and 16,700 carloads were moved. The powder was No. 1 giant and the amount used was 2.62 lbs. per hole, or 3 $^1/_3$ lbs. per cu. yd. There were 470 drills sharpened per week, although 1,100 drills were sharpened one week when in very hard rock. There were 105 men employed in and about the tunnel, working in three shifts. In 1877 the best month's advance was 388 ft., which was certainly an excellent record.

Musconetcong Tunnel.—Drinker describes the work on this tunnel in his book, also in *Trans. Am. Inst. Min. Eng.*, 1875. The tunnel is 12 miles from Easton, Pa., on the Lehigh Valley R. R., and was begun in 1872. The material penetrated was 3,731 ft. of very hard syenite gneiss, 460 ft. of limestone and 770 ft. of earth. One shaft was sunk. Top headings 8 x 26 ft. were first started by hand with a progress of 40 to 60 ft. per month. Ingersoll drills were subsequently installed and were operated on two carriages, each supporting three drills. The cutholes were 10½ ft. deep and started 9 ft. apart; the other holes were 12 ft. deep, starting 2¾ ins. diam. and ending up 1½ ins. diam. The 12 cut holes, drilled in pairs, were loaded with 25 lbs. of No. 1 dynamite and 50 lbs. of No. 2, and fired by electricity. No. 2 was used entirely in the squaring up holes. The total charge in the heading was 25 lbs. of No. 1 and 245 lbs. of No. 2, resulting in advance of 10 ft. for 408 ft. of drill holes. About 6 ft. of hole were drilled per cu. yd., and about 4 lbs. of No. 2 and 0.4 lb. of No. 1 were used per cu. yd. To drill these 408 ft. of holes required four 8-hr. shifts of six drills on each shift, so that each drill averaged only 17 ft. of hole per 8-hr. shift! The crew on each shift consisted of the following men, and I have assumed wages to be as given:

6 drillers,	at $3.00	$18.00
6 helpers	" 2.00	12.00
6 muckers	" 1.50	9.00
1 nipper	" 1.50	1.50
1 boss	" 3.50	3.50
Total per shift		$44.00

The bench was 12 ft. high and a row of four 12-ft. vertical holes was drilled 9 ft. back from the face; then two more 12-ft. vertical holes were drilled close up to the sides of the tunnel and 4½ ft. back of the face; then four bottom, horizontal holes, 10 ft. deep, were drilled into the face close to the floor of the tunnel. These 10 holes were charged with 107 lbs. of No. 2, and a 9-ft. advance was made by each firing. The bench was kept 500 ft. back of the face, and the bench crew consisted of 3 drillers, 3 helpers, 14 muckers, 1 nipper and 1 boss for each shift.

The total amount of explosives used in excavating 82,000 cu. yds. of heading and bench combined was 0.34 lbs. No. 1 plus 1.71 lbs. of No. 2 per cu. yd. of tunnel excavation, 1 exploder and 3 ft. of connecting wire per cu. yd.

The plant consisted of 30 drills, 4 compressors, 9 boilers, 2 machine shops, 1½ miles of 6-in. air and water pipe, 2 hoisting engines and boilers, pumps, etc. The coal and supplies were hauled one mile over rough roads with 4-horse teams, and there were 24 teams. In three years there were 27,000 tons of coal used. The force all told was about 1,000 men.

A Tunnel Through the Palisades, N. J.—In *Engineering News,* March 30, 1893, a brief description is given of a tunnel through the Palisades in N. J., about 2 miles north of the Weehawken Tunnel of the West Shore (see *Eng. News,* June 17, 1882). This tunnel, which was being built by the Hudson R. R. & Terminal Co., was begun Aug. 1, 1892, and is 5,070 ft. long, through trap rock. In solid rock the section is 27 ft. wide by 21 ft. high, the roof being an ellipse with a 9-ft. rise. The tunnel is driven by a top heading 7 x 18 ft., 24 holes 8 ft. long being drilled in the face; 8 center-cut holes and two rounds of 8 holes each are fired in three blasts.

Rand and Ingersoll drills with 2½-in. starting bits are used, and 30 ft. for each drill in a 10-hr. shift is considered good work. The dynamite is 60 per cent. A derrick car is used in mucking the benches which break out heavy; this car lifts the body of a mucking car from the track alongside and places the body close up to the bench to be filled. By this means large stones can be rolled into the car body. Wheelbarrow loads from the heading are dumped from staging into these car bodies. The derrick car also handles heavy stones.

The Tequixqueac Tunnel.—In *Engineering Record,* Feb. 29, 1896, an abstract is given of an article in *Engineering* on the Tequixqueac Tunnel, for drainage of City of Mexico. The tunnel is 6¼ miles long in sandstone; circular cross-section, 14 ft. diam. inside brick lining. The shaft-sinking method is outlined. Water was present in large quantities. Tunnel headings were 6½ ft. high, 5 ft. wide at bottom and 4 ft. at top, inside timbers, with a water ditch 3 ft. deep. Six holes 6½ ft. deep were drilled in the face and loaded with 6 lbs. of Nobel dynamite. Dynamite cartridges were dipped in grease. Drills were put into the holes to keep the charge down. Each blast threw down 8 cu. yds. of rock upon timber platform. Cars held 12 cu. ft., on 2-ft. gage, 12-lb. rails on iron ties, in 20-ft. lengths. No. 1 Root blower, 6-in. outlet, forced air through 6-in. spiral pipe to within 10 ft. of face, which was always sufficient for 2,100 ft. from shaft. Men worked under sub-contractor in 4-hr. shifts, then rested 4 hrs.; then worked 4 hrs. more, changing foremen every shift, which proved a splendid arrangement. There were 60 men on a heading shift thus: 2 timbermen, 4 drillers, 2 mucking debris, 22 loading cars and running them (2 pushing a car), 2 plate layers, 4 men timbering, 1 ventilator man, 10 men digging water channel with bars, 1 oiling and greasing, 2 hookers on, 2 dumpmen, overseers, foremen and time keepers. Each contractor was required to make a minimum progress in driving heading of 26 ft. in 24 hrs. As an actual fact, one heading was driven 43½ ft. in 24 hrs., and 728 ft. were driven in a month! So far as I know this record has never been equaled. It is stated that the organization of the forces was perfect, and that the working of 4-hr. shifts was a pronounced advantage, as the men worked with great energy even in very wet headings. It will be noted that each hand driller averaged not less than 13 ft. of hole (two 6½-ft. holes) in 8 hrs. It is not clear how 22 men could have been employed in so narrow a heading loading cars and tramming; certainly their yardage output per man was nothing remarkable, except for its lowness.

The Simplon Tunnel.—Americans may well study the methods by which this tunnel has been driven; for to have maintained a rate of speed practically double the rate usually made in good ground in this country is, of itself, sufficient to rivet the attention. In *Engineering News,* Aug. 30, 1900, is printed a valuable paper on this tunnel, by Mr. C. B. Fox, from which the following data have been abstracted: The Simplon Tunnel pierces the Alps between Italy and Switzerland, and its length when completed will be 64,725 ft., or nearly 11 miles. It will be completed in 1904. One full sized railway tunnel (No. 1) is being driven, and the heading of what will eventually be another tunnel (No. 2) is also being driven parallel with the main tunnel; the two headings being 56 ft. apart c. to c. The headings are bottom headings. Cross-headings connect the two, every 660 ft. The material is all carried to the portals, there being no shafts. The cross-section of tunnel No. 1 is 13.6 ft. at the formation level, increasing to 16.4 ft., with a total height of 18 ft. above the rails, giving an area of 250 sq. ft. The tunnel is lined throughout. Tunnel No. 2, which is being left as a heading, is 6.6 ft. high x 10.2 ft. wide, and is lined only where necessary. Cross-headings every 660 ft., making an angle of 56° with the axis of the main tunnel connect the two tunnels. The two parallel headings, each 59 sq. ft. in area, are driven by machines, the work being carried on day and night 7 days in the week. No. 1 is then enlarged to full size by hand drilling (one holding and one striking). No timbering is required in the north heading. Steel centers are used for placing the masonary arches. On the south headings a softer rock is encountered, requiring timbering. The excavated material is hauled, on a track having 2.6-ft. gage in 2-yd. cars drawn by a small locomotive designed with a very large boiler to avoid firing in the tunnel—a feature worthy of note.

The drilling machines are of the Brandt rotary type that cuts a core. A small four-wheeled truck carries a horizontal beam, the shorter arm of which supports the horizontal bar on which the drills are mounted, the longer arm being counterweighted. The bar is braced against the sides of the tunnel much as an ordinary column is held, except that, instead of screw jacks, a plunger operated by water pressure is used. Three or four machines are mounted on the bar. The drill is rotated by water, acting under pressure in two pistons, from which screw gearing conveys the motion to the drill rod. The total loss of power by friction is not over 30 per cent. The drill makes only 5 to 10 revolutions per min., according to the nature of the rock. The water for running each drill is 240 gals. per min. delivered under the enormous pressure of 1,470 lbs. per sq. in. The water enters the tunnel in wrought iron pipes, 3⅛ ins. inside diam.

and $^3/_{16}$ in. thick, made in lengths of 26 ft. In 3,750 ft. of tunnel the loss of pressure due to friction in the pipes is 147 lbs. per sq. in. The drill itself is a hollow pipe of tough steel, 2¾ ins. outside diam., provided with 3 or 4 sharp teeth, and it virtually chips or saws its way into the rock without pulverizing it as does a diamond drill. The machine has a feed of 2 ft. 2 ins., and when the drill reaches the end of the feed, it is pulled out, unscrewed and an extension tube is screwed on, the whole operation taking only 15 to 25 secs. (Note: This is a remarkably short time compared with the ordinary percussion drill.) The water after passing through the motors passes down through the drill tube and serves to wash out the chips and keep the bit cool. A hydraulic piston advances the machine and gives a pressure of 10 tons on the bit. The holes drilled are comparatively shallow, 4 ft. 7 ins. and 2¾ ins. diam. Three machine drills put down 10 to 12 of these holes in gneiss in 2½ hrs., or at the rate of 15 to 18 ft. per hour per drill; thus enabling an advance of 18 to 19½ ft. to be made in heading (59 sq. ft.) every 24 hrs. The record thus far made for a month's work is 682 ft. in the north heading, for December, 1903.

During the last three months of 1899 the work done was as follows: In the north headings there were three machines in tunnel No. 1 and two in the parallel heading of No. 2. The total distance driven in these two headings was 2,985 ft. in 89 days (24-hr.), or an average of 16.8 ft. per heading per day. The average cross-section of each heading was 57 sq. ft., so that a total of 6,409 cu. yds. was excavated. This required 507 attacks and 3,066 holes having a total depth of 26,600 ft., 14,700 re-sharpenings of the bits and 44,000 lbs. of dynamite. This is equivalent to 4.16 ft. of hole and nearly 7 lbs. of dynamite per cu. yd. The average period of a drilling attack was 2 hrs. 45 mins., while charging, firing and mucking took 6 hrs. 35 mins. There were 648 men and 29 horses on inside work, and 541 men on outside work.

On the south headings, where the rock is much harder, there were three machines in each of the two parallel headings. The total length excavated, with a cross-section of 62 sq. ft., was 2,880 ft., or 6,700 cu. yds. in 91 days, or 15.8 ft. advance in each heading every 24 hrs. This required 758 attacks, 7,940 holes with a total depth of 33,000 ft. and 56,000 lbs. of dynamite. Thus in hard gneiss nearly 5 ft. of hole, and 8½ lbs. of dynamite, were required per cu. yd., and each bit drilled 6½ ins. before re-sharpening. There were 496 men and 16 horses working in 8-hr. shifts on the inside work, and 346 men on outside work.

The organization of the work is as follows: Two men operate each drill, one feeding the bit forward, the other changing bits. There are three of these drill gangs in a heading, with one foreman and two men in reserve. Three or four holes are drilled in the center to a depth of 3¼ ft., and 7 or 8 holes are drilled around the outside of the face to a depth of 4.6 ft. The fuses of the center holes are brought together and cut off shorter than the others. The drill carriage is run back; a steel flooring is laid for a distance of 30 ft. back of the face and covered with debris to prevent damage to it by flying rocks. The blast is fired, and immediately afterward a valve is opened letting five jets of water play upon the rock to lay the dust. A truck is brought up, and four men clear a passage in front, two using picks and two using shovels, while on each side and behind are as many muckers as space will permit. The stone is thrown to both sides and into the car, shoveling being greatly facilitated by the steel flooring. The steel plates are taken up when cleared, and the car pushed forward. Finally the drill carriage is run forward, as soon as a way has been cleared, and the muck at the sides is removed at leisure during the drilling. The time consumed in an attack is:

Bringing up and adjusting drills	$^1/_3$ hr.
Drilling	1¾ to 2 hrs.
Charging and firing	¼ hr.

Clearing away muck	2 hrs.
Total	4½ to 5½ hrs.

This attack usually results in advance of 3¾ ft., or 18 ft. in 24 hrs.

A mechanical plow has been tried for clearing away the muck; water jets have also been tried, but thus far without success.

Air is forced in at the portal, which is kept closed by doors, but to carry away the smoke rapidly an 8-in. pipe (480 ft long) is lead up to within 45 ft. of the heading face, and air is driven through by a water jet. Each face receives 800 cu. ft. of fresh air per min. One jet of water ($^1/_{16}$ in. diam.) from the high pressure main supplies 1,000 cu. ft. per min. at the end of 480 ft. of 8-in. pipe. See *Engineering News,* May 27, 1897, for cross-section and plan. See also Aug. 13, 20, 27, 1903, for operating details.

Tunnel Near Peekskill, N. Y.—The following data are given in *Engineering News,* Dec. 17, 1903, by Mr. Geo. W. Lee, engineer for Sundstrom & Stratton, the contractors who built the double-track tunnel described. The tunnel is only 275 ft. long, and is on the line of the New York Central R. R., 2½ miles north of Peekskill. The yardage as shown on the plans was 7,028 cu. yds., but as the rock lay in strata dipping at an angle of 45°, it broke out on the uphill side so as to leave large pockets, in consequence of which the contractor took out 10 per cent. more rock than he was paid for. Owing to the seamy condition of the rock, and the proximity of the tunnel to the main line traffic, very light charges of dynamite were used, which increased the cost and delayed the progress. Rand steam drills, 3-in., were used. A heading 8 x 10 ft. was run and the bench was kept close behind. Rock from the heading was removed in small narrow gage cars; rock from the bench was loaded into standard gage cars by derrick cars. The following was the cost of the tunnel excavation:

Equipment (less present value), supplies and repairs	$2,893.52
Dynamite and exploders	1,604.58
Coal	570.80
Oil, waste, etc.	92.80
Lumber for houses and shops	129.88
Miscellaneous	92.10
Labor	22,212.86
Total	$27,596.54
Average cost per cu. yd. paid for	3.93
" " " " " taken out	3.54

The tunnel was lined with 1:2:4 concrete; 692 cu. yds. in the bench walls; 932 cu. yds. in the arch; the portal head walls were of 1:3:6 concrete, 324 cu. yds. The cost of the concrete was as follows for the 1,948 cu. yds.:

Cement at $1.63 per bbl.		$5,755.50
Sand at 75 cts. per cu. yd.		662.94
Crushed stone at 80 cts. per cu. yd.		1,303.20
Lumber:		
Mixing platforms and runways	$336.89	
Ribs, including hand sawing	234.10	
Backing boards	134.44	
Lagging	341.04	
Sheathing	268.49	
Plates, sills, studs and braces	182.75	
		1,497.71
Coal		118.73
Oil		16.12
Hardware, nails, spikes, etc.		224.39
Tools		181.10
Freight on stone, cement, etc.		3,089.86
Labor, including supt., foreman, etc.		8,036.31
Total, $10.72 per cu. yd.		$20,885.86

In the approaches to the tunnel and in widening cuts south of the tunnel 45,698 cu. yds. of rock were removed. On account of proximity to traffic, blasting could be done only at limited periods, which made the cost of excavation high. Rock was loaded on flat cars with stiff leg derricks provided with bull wheels. The cost was as follows:

Equipment (less present value), supplies and repairs	$11,673.60
Dynamite and exploders	6,588.82
Coal	2,490.13
Oil, waste, etc.	370.59
Lumber for buildings	634.22
Miscellaneous	373.19
Labor	69,550.66
Total	$91,681.21
Average cost per cu. yd. paid for	2.24
" " " " " taken out	2.01

Cost of Lining Tunnels.—(The data on the tunnel near Peekskill, just described, should be consulted for data on concrete lining.) Drinker gives the following data on the lining of Carr's Tunnel (825 ft.) on the Pennsylvania R. R. in 1868-1869. Brickwork: 609,000 brick in the arch (5 per cent. broken and lost); 10.44 bushels of neat cement (no sand used in the mortar) laid 1,000 bricks, the mortar forming 30 per cent. of the brick masonry; the arch was 25 ins. thick, 24½-ft. span and 9-ft. rise:

	Cost per M.
Bricks f. o. b.	$8.80
Loss in handling	.51
Unloading and delivering	1.92
Laying	5.84
Cement	5.10
	$22.17

Bricklayers received 40 cts. per hr.; helpers, 17½ cts. per hr.; carpenters, 27½ cts. per hr.; laborers, 17 cts. per hr.

Stonework: 1,730 perches (25 cu. ft.) of rough masonry for side walls, presumably sandstone; 187 perches of ring stone; 25 perches wasted in dressing. The bench walls were 4 ft. wide at the bottom, 3 ft. at the top and 13 ft. high:

	Cost per perch.
Quarrying (1,730 perches)	$4.80
Cutting (1,730 perches)	4.36
Hauling (1,942 perches)	1.06
Handling and laying (1,917 perches)	2.80
Cement, 1.65 bu. per perch (8 $^1/_6$ per cent. of the masonry	.81
Total	$13.83

Stone cutters and masons received 35 cts. per hr.; quarrymen, 17½ cts.; laborers, 17 cts. The stone side walls were laid in 8 courses averaging 2 ft. thick each; hence there were 52,800 sq. ft. of beds cut; and estimating each stone 3 ft. long and dressed for 1½ ft. back of the face on joints, there were 14,300 sq. ft. of joints; making a total of 67,100 sq. ft. of cutting which cost 11.2 cts. per sq. ft. This is said to have been too high a cost, if the measurements were correct.

Arch centering cost $1,400, to which was added $600 for moving the centering forward from time to time; making $2.40 per lin. ft. of tunnel, to which must be added $0.70 per ft. for scaffolding.

In *Engineering News,* Oct. 25, 1894, is given an abstract of a paper read before the Montana Society of C. E.'s, by Mr. H. C. Relf, on the lining of the Mullan Tunnel on the Northern Pacific Ry. with masonry to replace timber. The tunnel is 3,850 ft. long, 20 miles west of Helena. Falls of rock and fires in the tunnel had caused numerous delays. The original timbering consisted of sets 4 ft. c. to c. of 12 x

12-in. timbers, with 4-in. lagging. The size was 16 x 20 ft. in the clear.

Concrete side walls (30-in.) and four-ring brick arch were built in place of the old timbering. A 7-ft. section was first prepared by removing one post and supporting the arch by struts. Two temporary posts were set up and fastened by hook bolts; and a lagging was placed back of them to make forms to hold the concrete. Several of these 7-ft. sections were prepared at a time, each two being separated by a 5-ft. section of the old timbering. The mortar car delivered Portland cement mortar (1 to 3) through a chute, making an 8-in. layer of mortar into which broken stone was shoveled until all the mortar was taken up by the stone voids. In 10 to 14 days the walls were hard enough to support the arches which were then allowed to rest on the walls, and the posts of the remaining 5-ft. sections were removed, and concrete placed as before. About 4 parts of mortar were used to 5 parts of broken stone, which is a very rich concrete. The average progress per working day was 30 ft. of side wall, or 45 cu. yds.; and the average cost, including removal of old timber, train service, engineering, superintendence and interest on plant, was $8 per cu. yd. of concrete wall. From 3 to 9 ft. of brick arch were put in at a time, depending upon the nature of the ground. To remove the old timber arch, one of the segments was partly sawed through, and a small charge of dynamite exploded in it; the debris being caught on a platform car, from which it was removed to another car and conveyed away. The center was then placed, and the cement car used to mix mortar on. Brick were 2½ x 2½ x 9 ins., four ringings, making a 20-in. arch and giving 1.62 cu. yds. per lin. ft. of tunnel. The bricks were laid in rowlock bond. Two gangs of 3 bricklayers and 6 helpers each, laid 12 lin. ft., or 19.4 cu. yds., of brick arch per day. The brick work cost $17 per cu. yd., making the total cost of tunnel lining $50 per lin. ft. The work was still in progress at the time of writing.

CHAPTER XVII.

COST OF DRIFTING, SHAFT SINKING AND STOPING.

Definitions:

Adit, a small tunnel driven from the surface into a hill.

Breast, the face or working end of an adit, drift or stope.

Cross-cut, a small, underground, horizontal opening driven across the trend of the vein or formation.

Development, or Dead Work, the shafts, drifts, cross-cuts and other openings made preparatory to stoping the ore.

Drift, a small, underground opening that follows the direction of the vein or lode.

Incline, a shaft-like opening extending downward from the surface at an angle of less than 90°.

Level, a horizontal opening in a mine connecting with a shaft or incline. Levels are usually 100 ft. apart, occasionally 150 ft.

Raise, or Upraise, an opening driven up from one level to another.

Shaft, a vertical opening extending from the surface, and used as an entrance and exit.

Stoping, mining the mass of ore between levels.

Winze, a small opening connecting one level with another, as for ventilation.

General Considerations.—The reader is supposed to be tolerably familiar with one or more books on mining, such as Ihlseng's, or Foster's, but a few remarks may help to a clearer understanding of the reasons why underground work is so much more expensive than open-cut work. As we have seen in the last chapter, wherever a narrow face in a tunnel is exposed to attack by blasting, the explosive has a great deal of work to do per unit of excavation, because of the great area per unit that must be sheared off. To place the larger amount of explosive required means either the drilling of drill holes having a large diameter, or the drilling of holes close together. Miners in the days of hand drilling were obviously compelled to choose the latter method; and, even since the introduction of dynamite and power drills, the practice has continued to be close spacing. I am not at all sure, however, that this is the best practice; and I look to see more powerful drills used in the future for drilling holes of greater diameter, spacing the holes farther apart. The advantage of this method has been proved in subaqueous excavation, and there is every reason for believing that larger holes will prove more economic in underground work.

In underground work, since derricks cannot be used, it is necessary to break up the rock to sizes that will require little or no sledging, mudcapping or blockholing, before loading by hand into cars. This means likewise either close spacing of drill holes, or drill holes of large diameter. We see, therefore, that in any case more work in drilling and more explosives are required to break rock underground than in open cuts. In addition to these greater costs are the costs of timbering, or filling, of pumping, of lighting, of hoisting, of greater delays after blasting, of ventilation, etc. Finally, the miners' unions have gradually raised rates of wages to a point that makes comparison of open-cut work with underground work impossible, if the dollar is the unit of comparison.

Shaft sinking is well known to be more expensive than tunneling, ton for ton, or yard for yard; and it is impossible to sink a shaft with anything like as great rapidity as a tunnel of equal size can be driven. The chief reason for the slowness of shaft sinking lies in the fact that the greater part of the muck must be removed after each blast before drilling can begin; whereas in a tunnel the drillers can begin while the muckers are still at work. Timbering a shaft also delays excavation and hoisting to a far greater degree than tunnel timbering. It is harder to ventilate a shaft, and obviously harder to drain a shaft than a tunnel. The fixed expense of a hoisting plant and engineer is out of all proportion to the small amount of material hoisted daily. All these factors, and certain others of less importance, make shaft sinking slower and more expensive than tunneling. The number and size of drill holes and the amount of explosives are practically the same in shafts and tunnels of equal areas. As the depth of the shaft increases, there comes a time when work is retarded by the speed and capacity of the hoisting plant. The best record of speed of shaft sinking for moderate depths is that given by Mr. Edward J. Way, in the *Trans. Am. Inst. Min. Eng.,* Feb., 1904. A shaft, 6 x 21 ft., was sunk 858 ft. in five months; during the last month, May, 1903, the record was 213½ ft. In all there were 4,032 holes drilled to an average depth of 7 ft. 2 ins.; 418 cases of gelatine, 2,475 coils of fuse, 86 boxes of detonators and 207 boxes of candles were used. This work was done in South Africa, and the costs are given, but unfortunately Mr. May does not give the organization of the force nor the rates of wages.

The following examples of cost will give a fair idea of the range; but I trust that some of my readers will be kind enough to send me other detailed examples, stating conditions, so that future editions of this work may be of greater value both to the experienced as well as the inexperienced mining man.

Cost of Tunneling Melones Mine.—In *Trans. Am. Inst. Min. Eng.,* 1898, Mr. W. C. Ralston gives data of cost of driving an adit at the Melones Mine, Calveras Co., Cal., in 1898. The work consisted in extending a tunnel that was already 1,080 ft. long. The tunnel was 7 x 8 ft. in the clear; grade, ¼ per cent.; 12-lb. rails; 22-in. gage; ties, 4 x 6 ins., 3 ft. c. to c.; walking plank, 2 x 20 ins. An Ingersoll-Sergeant compressor, class "B," driven by a 5-ft. Pelton water wheel, delivered air to an 8-in. pipe at a pressure of 200 lbs. When water was scarce a Nagle engine, 12 x 16, was used. A No. 4½ Baker blower, run as an exhaust, sucked air through a 11-in. pipe.

In 255 days only 2 were lost. The average progress was

10.22 ft. per day, or 306.9 ft. per month, working from one face. The best run for two consecutive weeks was 184 ft., or 92 ft. per week. The rock was firm greenstone (diabase), brown slate and talc schists filled with quartz stringers. Only nine sets of timbers were used. The work was rushed, and no especial effort was made to economize. Three 8-hr. shifts of 7 men each (4 drillers and 3 muckers) were worked. Two 12-hr. shifts were worked, each with a team and driver and an engineman. One 10-hr. shift was worked with a blacksmith and helper, a mechanic and an outside-man. Total, 29 men. Two Ingersoll drills, 3⅛-in., were used, and the repairs on them were only $91 for over 2,600 ft. of tunnel.

MELONES MINE (1898).

Actual cost (exclusive of management) of 2,608.5 ft. of tunnel.

		Cost per Lin. Ft.
Labor (29 men)	$19,501.46	$7.47
Powder, 2,000 lbs. No. 1 Hercules @ 16.6 cts. Powder, 25,500 lbs. No. 2 Hercules (40%) @ 11.9 cts.	3,405.65	1.30
Fuse, 74,000 ft. @ 51.7 cts. per 100 Caps, 200 boxes @ 60 cts. per 100	500.20	.19
Wood, 333½ cords @ $5	1,667.50	.63
Water, 15 cts. per inch	828.50	.32
Coal, 11,591 lbs. Cumberland @ $15	179.43	.06
Foot plank, ties, and 9 sets of timbers, 8,466 ft. B. M. @ $20 per M	169.32	.06
Candles, 3,040 lbs. @ 7½ cts	262.04	.10
Steel rails, 21,555 lbs., 1¼ and 2¾ cts	567.62	.22
Air pipe, 11-in., 18 cts. and 30 cts. Air pipe, 3-in., 22 cts. Water pipe, 2-in., 11¼ cts	1,042.45	.45
Horse feed: hay, 1½ cts.; barley, .019 ct	267.16	.10
Steel, drill parts, oil and tools, etc	316.92	.12
Total	$28,708.25	$11.02

N. B.—The air and water pipes used in running different cross cuts were not left in place, but were moved as needed. Of the 21 miners, six received $3 a day, the rest $2.50 a day, and carmen $2 a day. Powder was used liberally to pulverize the ore.

Cost of Tunneling, Hogsback Mine.—At the Hogsback Mine, Placer Co., Cal., the force was 21 men divided into 3 8-hr. shifts of 5 men each; two 12-hr. shifts of an engineer, driver and horse on each shift; and two 10-hr. shifts of blacksmith. The average hardness was the same as at the Melones Mine, but the blocky nature of the ground permitted more effective blasting, and only half as many holes were drilled.

HOGSBACK MINE (1888).

Actual cost (exclusive of management) of 1,559.6 ft. of tunnel, 7 × 8 ft.

		Cost per Ft.
Labor	$12,131.49	$7.77
Powder, 10,021 lbs. No. 2 @ 14¾ cts.	1,478.10	.90
Fuse, 23,045 ft. @ 54½ cts., caps @ 80 cts.	165.59	.10
Wood, 522 cords @ $2.75	1,435.50	.92
Charcoal, 1,580 bu. @ 20 cts.	316.00	.20
Candles, 1,755 lbs. @ 13¼ cts.	232.53	.14
Gang plank and ties, 7,624 ft. B. M. @ $22.50	171.54	.10
Timbers, 21 sets @ $1.80	37.80	.02
Steel rails, etc. (16-lb.), 20,048 lbs. @ 4 cts.	801.92	.51
Air and water pipes, 3-in. @ 29½ cts.; 1-in. @ 6¼ cts.	761.43	.48
Horse feed: hay, 3 cts.; barley, 3 cts. per lb.	349.60	.22
Steel, drill parts, oil, tools, etc.	916.33	.58
Total	$18,797.83	$11.94

N. B.—Each man received $3 a day.

COMPARISON OF BEST WEEK'S RECORD IN EACH MINE.

	Melones.	Hogsback.
No. of men	31	20
No. of holes drilled, 5 ft. each	291*	150†
Holes reblasted	6	11
Time used in drilling	45 hrs.*	26 hrs.†
Average time drilling per shift	2⅛ hrs.	1¼ hrs.
Powder, lbs.	925	344
Candles, lbs.	77	72
Wood consumed, cords	None.	21

* 6.47 ft. of hole per hr. of actual drilling.
† 5.77 ft. of hole per hr. of actual drilling.

Rock extracted per shift, cars..........	23.19	21.8
Rock extracted for week, 3 shifts, 7 days.	487 cars.	458
Progress for week, ft..................	92	73.6
Cu. yds. solid rock (7 × 8)............	191	149
Previously reported	2,310	404
Labor expenses for week...............	$551.75	$435.75

Observe that a 7 x 8-ft. tunnel has 2.08 cu. yds. solid rock per running ft., and taking the average runs at the two mines we have:

	Per Cu. Yd. Rock in Place.	
	Melones.	Hogsback.
Powder, No. 2, 40% Hercules.......	5¼ lbs.	3⅛ lbs.
Fuse	13½ ft.	7 ft.
Labor (exclusive of management)....	$3.60	$3.70
Ft. of hole..........................	7.6	5

Observe that each car held about $^4/_{10}$ cu. yd. solid rock at Melones and $^1/_3$ cu. yd. at Hogsback.

Cost of Sinking and Stoping at the Utica Angels.—In *Trans. California Miners' Assoc.*, 1899, Mr. J. H. Collier, Jr., gives the following data on mining at the Utica Angels, Calaveras county, Cal.: The mine is in a mineralized zone of crushed diabase more or less altered to schist, being in the celebrated "Mother Lode." The ore is a nearly vertical quartz vein carrying free gold and pyrites. The method of timbering and stowing or filling is used, no opening over 16 ft. high being left. Waste from prospect cross-cuts is used for stowing.

In stoping two 10-hr. shifts of 50 men per shift, supply a 60-stamp mill with 300 tons of ore per day. The cost of labor in stoping is as follows per shift:

12 miners, at $3	$36.00
12 helpers, at $2.50	30.00
15 shovelers, at $2.50	37.50
6 men tramming, at $2.50	15.00
5 timbermen, at $3	15.00
Total labor stoping 150 tons	$133.50

The ore is loaded for hoisting into skips from chutes.

In shaft sinking, after removing the surface soil, men begin with churn drills, having a 2-in. bit to start with, and tapering to 1¼ ins. at a depth of 6 or 7 ft. After reaching solid rock four air drills are used, two at each end of the shaft on bars about 3 ft. above the bottom. Eight rows of holes are drilled across the short way of the shaft, three holes to a row, 6 to 7 ft. deep; six holes making the V-cut. Each hole is charged with 6 to 9 sticks (1⅛ x 8 ins.) of 40 per cent. dynamite, and the firing is done with a battery. The force consists of 6 machine men and 6 helpers, divided in three 8-hr. shifts; 2 enginemen on 12-hr. shifts; 2 machine men with 2 helpers and 3 timber men, who work only when machines run. Each man receives $3 per shift. A three-compartment shaft (7 x 16 ft. outside of timbers), compartments 4 ⅓ x 5 ft. each, was sunk 825 ft. in 7 months. The timber sets are 5 ft. apart and 2-in. lining is used. The author gives a good description of framing and placing the timbers.

Drifts are timbered with two 7-ft. legs set 6 ft. apart at the bottom and a 5-ft. cap on the top. Machine drills put in three horizontal rows of holes, four holes in a row, the center holes forming a vertical V-cut. In talc and mica-schist the drill clogs badly on down holes. In one case a 7-ft. dry (up) hole was drilled in ½ hr., whereas it took 2 hrs. to drill a 7 ft. down (wet) hole. A jet of water under pressure enables the drills to cut much faster on down holes.

Cost of Sinking and Drifting at the Lincoln Gold Mine.—The cost of sinking the Lincoln shaft at Sutter Creek, Amador county, Cal., is given in detail in the fifth annual report of Mr. E. C. Voorheis, Superintendent of the Lincoln Gold Mine Development Co. Mr. Voorheis has given me certain additional data not contained in the regular report, as follows:

In shaft sinking the men work 8-hr. shifts, but in drifting 10-hr. shifts. The average depth of each drill hole in shaft sinking is about 6 ft., but in drifting 5 ft. The drilling was done by power, using the "Baby" giant drill manufactured by the Compressed Air Machinery Co., of San Francisco. The powder used was Herclues, 40 per cent. nitroglycerin, and the fuel for steaming was crude oil delivered at the mine at $1.50 per bbl., of 42 gallons.

The work of sinking the Lincoln shaft was completed May 24, 1902, sinking from a level of 1,260 down to the 2,000-ft. level, a distance of 740 ft.

The size of the excavation is 8 x 17 ft. Material encountered in sinking, greenstone and hard black slate.

The labor cost of sinking and putting in timbers was as follows: It required 3,864 blasting holes drilled in the bottom of the shaft, or 5.2 per foot of shaft, which took:

2,956 days' labor at $2.75 per 8-hr. day		$7,129.00
1 day foreman, 350 days, at $4		1,400.00
1 night foreman, 282 days, at $3.25		916.50
Total labor cost of sinking 740 ft. and putting in timbers		$9,445.50
Labor cost per ft. of sinking 740-ft. shaft..	$12.76	
12,450 lbs. Hercules powder used, costing		1,307.25
Cost of powder per ft. of shaft	1.77	
Amt. of powder per ft. of shaft, 16.8 lbs.—		
35,800 ft. of fuse used, costing		125.30
Cost of fuse per foot of shaft	.17	
Amt. fuse per ft. of shaft, 48.4 ft.—		
46 boxes of Lion caps were used, costing..		46.00
Cost of caps per ft. of shaft	.06	
2,400 lbs. of candles used, costing		288.00
Cost of candles per ft. of shaft	.39	
Amt. candles used per ft. of shaft, 3.2 lbs.—		
148 sets of timbers, requiring 207,200 ft. of lumber; average cost, $18 per M.		3,729.60
Cost of lumber per ft. of shaft	5.04	
Amt. lumber per ft. of shaft, 280 ft.—		
Total cost, labor, lumber, light, etc. ..		$14,941.65
Cost per ft. for labor, lumber, light, etc.	$20.19	

Total cost, labor for engineers, blacksmiths, framing timbers, skip tenders..		$6,224.00
Cost of top expense per ft. of shaft	$8.41	
Total cost for fuel for sinking 740 ft.		5,893.50
Cost of fuel per ft. in sinking 740 ft.	7.96	
Total cost of sinking shaft, including all expenses except office expense..		$27,059.15
Total cost per ft. of shaft.	$36.56	

During the time the shaft was being sunk 60,025 tons of water were hoisted, or 14,406,200 gallons. 9,456 tons of waste were hoisted from the bottom of the shaft to the surface, which added to the water would make a total of 69,481 tons hoisted to the surface during the time the shaft was being sunk.

On July 1 cross-cutting for the vein channel was begun, starting from the north side of the shaft and gradually turning to the west, until a point at right angles to the foot wall of the shaft, which is at right angles to the formation of the ground, was reached. This cross-cut has been extended in a westerly direction 642 ft. At a point 187 ft. west of the foot wall of the shaft the cross-cut passed through a fissure of soft black slate, 40 to 60 ft. wide, carrying a good deal of water on the hanging wall side and some stringers of quartz, which was supposed to be the channel that carried the ore bodies on the upper levels. We drove north on the foot-wall side of this fissure 233 ft. and south 250 ft., making in all 483 ft. driven on the channel. At the end of each drift we cross-cut both east and west 12½ ft. each way, which is room enough to run cross-cuts with the diamond drill.

The following is a detailed statement of the cost of running 1,175 ft. of drifts and cross-cuts on the 1,950-ft. level. Most of the rock was hard greenstone; average size of tunnel, 5 x 8 ft.:

1,428 days' labor, *		$3,772.50
168 days for day foreman at $4		672.00
134 days for night foreman at $3.25		435.50
Total labor cost for drifting and cross-cutting 1,175 ft.		$4,880.00
Average labor cost per ft.	$4.153	
11,150 lbs. of powder used, costing		1,226.50
Average cost of powder per ft.	1.043	
26,500 ft. of fuse used, costing		79.50
Average cost of fuse per ft.	.068	
35 boxes of Lion caps used, costing		35.00
Average cost of caps per ft.	.03	
800 lbs. of candles used, costing		96.00
Average cost per ft. for candles	.082	
Total cost per ft.	$5.376	
Total cost		$6,317.00

3,258 blasting holes were drilled. Average number of holes drilled per ft., 2.77.

The average amount of powder used for each blasting hole was 3.42 lbs. of 40 per cent. Hercules.

Average amount of fuse per ft., 22 ft.; average number of caps per ft., 3; average amount of candles per ft., 0.68 lb.

Cost of the Newhouse Tunnel.—In the *Engineering and Mining Journal,* April 19, 1902, Mr. H. Foster Bain gives the following: The Newhouse tunnel starts at Idaho Springs, Col., at an altitude of 7,543 ft. It is a 12 x 12-ft. excavation now 12,861 ft. long, and eventally will be 5 miles long. In Feb., 1902, the average progress was 9.2 ft. per day from one face, or 267.6 ft. for the month. The rock never breaks beyond the holes; it is a granite-gneiss and schist dipping away from the face. Twenty holes, 10 ft. deep, are drilled, 8 cut holes, 1 plunger hole at the upper center of the cut, 3 back holes along the roof, looking up. and 4 side holes on each side looking a little out and up.

* Miners at $2.75 per 10-hr. day and car-men at $2.50.

Three Leyner-Water drills are used; two 3-in. sluggers on two columns drilling the side and cut holes, and one Model 5 drill mounted on a bar drilling the plunger hole and the back holes. The drill crew consists of 5 men, who work from 7 A. M. to about 5 P. M.; they do not stop drilling at noon. At 1 P. M. a powder gang of 5 men comes on and attends to track laying, timbering, putting in water box, etc., until the drilling is done. They lay the permanent track, which is kept 250 ft. from the face; the temporary mucking track being run up the drilling platform. The tracks are 18-in. gage, 35-lb. rails, on 4 x 8-in. x 8-ft. ties. The water box is 1 x 2 ft., made of 2-in. stuff and is carried under the track. The powder gang tears down the drills, covers the floor with ⅝-in. steel plates in sections 6 ft. square for 40 ft. back of the face, and then fires the cut holes by electricity; this takes 40 mins. The cut is fired 2 to 5 times with 60 per cent. gelatin powder, 8 to 9 sticks being used for the first round. In all about 100 lbs. are used in the cut holes. The side and back holes are fired with 40 per cent. powder, 50 to 70 lbs. being used. The holes are not tamped, and the blasting takes 2½ to 3½ hrs. During the blasting compressed air is delivered to the face through a 4-in. pipe, and drives the smoke back to a 19-in. exhaust pipe laid between the rails on top of the water box. After blasting, the gang cleans up the track ready for the muckers and loads the first train of cars.

At 10 P. M. a mucking gang of 6 men goes on; the boss picks down, and three men shovel while two rest. The cars (3 x 3½ x 5 ft.) hold 35 cu. ft., and 45 to 60 cars are filled per night. Mules have been used for tramming, but a 16-H. P. Baldwin-Westinghouse electric locomotive has just been installed. In all there are 32 men employed on company work, including blacksmiths.

The power plant includes three 80-H. P. boilers, of which two are in use at one time; two 16 x 16-in. two-stage Norwalk compressors; one 22 x 24-in. two-stage compressor;

one No. 7 Root blower driven by a 50-H. P. Atlas engine; 30-H. P. engine driving the 20-kilowatt, 500-volt dynamo. For drilling the air is used at 160 lbs. pressure, having been gradually worked up from 110 lbs.

Each of the drill crew receives \$3 a day plus \$1 on an average for bonus. The total bonus is \$6 per ft. for all over 160 ft. per month, or about \$500, and is money well expended, the men working long hours and hard rather than lose a round.

The detailed cost per foot is shown by the following abstract from the manager's report for the year ending Aug. 31, 1900:

Labor:		
Drill crews and foremen		\$3.01
Trammers, blasters and drivers		4.34
Blacksmith shop		1.03
Engineers		1.15
Explosives		4.55
Oil and waste		.12
Coal		3.91
Mules, feeding and shoeing		.32
Drill repairs		.82
Premiums		1.87
Tools		.33
Timber		.16
Track and material		1.84
Track—labor and repairs		2.07
Engineering and surveying		.22
Salaries and office expenses		1.97
Miscellaneous—		
Legal expense	\$0.07	
Insurance	.11	
Taxes	.12	
Minor items	.73	
		1.03
		\$28.74

Total, \$79,478 for driving 2,959½ feet or \$28.80 per foot. The previous year this cost was \$27.74 and the actual breaking cost was \$19.68. The increase was due to the increased cost of supplies and the greater expense of working in a larger tunnel.

Cost of Sinking and Drifting at the Homestake Mine.—In *Mining and Scientific Press,* Feb. 13 to Mar. 12, 1904, Mr. Bruce C. Yates gives data on the methods and costs of mining work in the Homestake Mine, Lead, S. D.

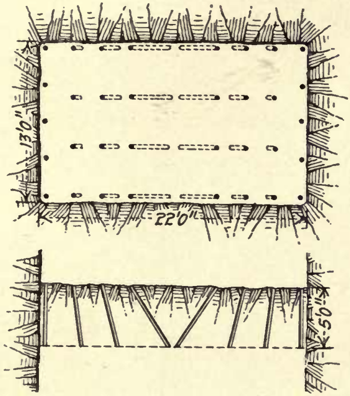


Fig. 54.

Fig. 54 shows the location of drill holes in shaft sinking. The rock is a hard carbonaceous slate dipping at an angle of 60° to 90° from the horizontal, which makes blasting the cut quite difficult. When sinking is resumed in any of the shafts, a 25-H. P. engine, operated by reheated compressed air, is installed on the lowest level. Miners in the shaft usually work under contract. At the Ellison shaft, which has three compartmnts, two of which are 5 x 10 ft., and the third 6 x 10 ft. in the clear, the force employed on each shift is: 4 miners (contractors), 1 engineman, 1 lander, 1 timberman and 1 carpenter working 2½ shifts framing one set of timbers which provide for 6 ft. of shaft. The actual time in placing one set of timbers is 8 hrs., but the timbermen are kept busy one shift in each 24 hrs. running timber from the saw mill and putting in extra bracing. The length of a shift is 10 hrs., but, as the men are lowered twice each shift on company time, the actual working time is about 9 hrs.

The cost of labor in sinking 1 ft. is:

4 miners,	2 shifts, at $4	$32.00
1 timberman,	1 " " 4	4.00
1 engineman,	2 " " 4	8.00
1 lander,	2 " " 3	6.00
Total labor cost per ft.		$50.00

To this must be added the cost of timber sets (12 x 12-in. timbers), as follows:

Timber for 1 set, 1,500 ft. B. M.		$26.85
Framing 1 set		10.00
4 miners,	one shift, $4	16.00
1 lander	" " 3	3.00
1 timberman	" " 4	4.00
1 engineman	" " 4	4.00
Total cost per set		$63.85

Cost per ft. of shaft, $10.64. The costs of the lining, of explosives and of power are not included in the foregoing.

There are three sizes of drifts: 6 x 7 ft. for prospect drifts; 7 x 8 ft. for single-track working drifts; 12 x 8 ft. for double-track working drifts. Figs. 55 and 56 show two methods of locating holes in single track drifts; holes are 6 ft. deep, and the advance is 5 ft. Two machines are used for drilling, preferably mounted on a horizontal bar. The bar set up is preferred to the column set up, because the machines can be set up immediately after blasting, and because the sides of the drift are carried along much more evenly. The following shows the cost of driving 5 ft. of 7 x 8-ft. drift:

1 miner drilling,	2 shifts, at	$3.50	$7.00
1 helper "	2 " "	3.00	6.00
1 miner blasting,	1 " "	3.50	3.50
1 helper "	1 " "	3.00	3.00
1 shoveler	2 " "	3.00	6.00
Explosives, 40 p. c. dyn., cap and fuse			10.75
Blacksmith, repairs of drill and mch. shop			4.85
Total cost of 5 ft.			$41.10
Cost per ft.			8.23
Cost per ton			1.47

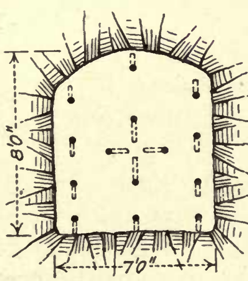


Fig. 55.

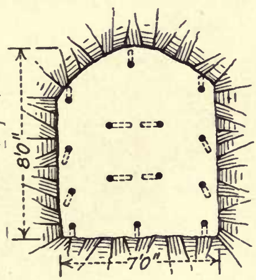


Fig. 56.

The above does not include track laying or putting in air pipe. There is no timbering in drifts. Dynamite costs about 12 cts. per lb. The drills used are 3 or 3⅛ ins. When two drills are worked, one helper serves two drills, and the progress is nearly doubled. The cars have a capacity of 20 cu. ft. each, and hold 10 cu. ft. of solid rock, weighing 1 ton. Hauling is done with horses at a cost of 0.3 ct. per ton per 1,000 ft.

The cost of driving a double-track 8 x 12 ft., drift, requiring 20 holes 6 ft. deep, giving an advance of 5 ft., is as follows:

1 miner drilling,	3	shifts, at	$3.50	$10.50
1 helper "	3	" "	3.00	9.00
1 miner blasting,	1	" "	3.50	3.50
1 helper "	1	" "	3.00	3.00
1 shoveler	3	" "	3.00	9.00
Explosives, caps and fuse				16.90
Blacksmith, repairs and mch. shop				6.50
Total cost of driving 5 ft.				$58.40
Cost per ft.				11.68
Cost per ton				1.22

Cross cuts and drifts in ore are from 18 to 24 ft. wide and 10 to 12 ft. high.

In making a "raise," 6 x 6 ft., one machine driller and helper will make 100 ft. of raise in 60 shifts, provided they are not required to "car" the rock; the location of holes is similar to that in drifting.

Cost of Drifting and Stoping at Cripple Creek.—In *The Engineering and Mining Journal,* Nov. 21, 1903, Mr. J. R. Finlay gives the following data: At the larger mines of Cripple Creek the cost of mining the total product of ore and waste is from $2.50 to $3.50 per ton, which includes all the expenses of the company. The above cost includes sorting the ore. The ore occurs in small veins, the largest and best of which are in granite. In one of the large mines the work done during one month was as follows: 18,910 tons were mined from 40 different stopes, but including the development work (drifts, cross-cuts, etc.) 24,931 tons were mined at a cost of $2.55 per ton, and this was reduced by sorting to 7,093 tons of shipping ore, making the cost $8.96 per shipping ton. Wages average $3.40 for 8 hrs.; coal, $4.60 per ton; timber, $20 per M. Part of the rock is ordinary, unaltered granite, not excessively hard, but not soft;

and part is equally hard porphyry. The costs in detail were as follows:

	18,910 Tons. Stoping, per Ton.	896 Ft. Drifts, per Ft.	1,229 Ft. Crosscuts, per Ft.	112 Ft. Raises and Winzes, per Ft.
Machine men and helpers and hand miners	$0.34	$1.86	$2.01	$3.16
Trammers, shovelers, pipe and track men, etc.	0.41	1.03	1.04	0.98
Timbermen	0.23	....	....	1.19
Total underground labor...	$0.98	$2.89	$3.05	$5.33
Cost of machines, compressed air, drill sharpening, repairs, etc..	0.15	0.97	1.05	1.66
General tramming cost, repairs on cars, oil, supplies, etc.	0.03	0.07	0.09	0.06
Explosives	0.13	1.43	1.66	1.42
Lumber and timber	0.28	....	....	1 52
Hoisting and tramming on surface	0.23	0.46	0.53	0.40
General expense, bosses, assaying, surveying, office, etc.	0.23	0.58	0.67	0.45
Supplies, miscellaneous	0.04	....	....	....
Total cost	$2.07	$6.40	$7.05	$10.84

In *Mining and Scientific Press,* Feb. 13, 1904, an abstract is given of the annual report of Mr. F. M. Kurie, assistant general manager of the Portland Gold Mining Co., Cripple Creek, Col. The report gives the following tables of costs for development work and stoping for the year 1903, which is the lowest in the history of the company:

	Per foot.		
Breaking	19,808 Ft. Drifts and Crosscuts.	1,451 Ft. Raises and Winzes.	Stoping per ton.
Machinemen and miners....	$1.969	$3.458	$0.369

Shovelers ...	$0.038		$0.193		$0.218	
Air and air drills.	1.049		2.02		.179	
Explosives	1.444—	$4.50	1.617—	$7.288	1.39—	$0.905
Tramming—						
Trammers	.966		.883		.175	
Pipe and trackmen	.124		.069		.008	
General tramming expense	.098—	1.188	.103—	1.055	.034—	.217
Timbering—						
Timbermen	.029		1.503		.215	
Lumber and timbers	.017—	.046	.947—	2.45	.310—	.525
Hoisting		.583		.604		.254
General expense ...		.70		.735		.291
Total average cost......		$7.017		$12.132		$2.192

In discussing the most economic methods of drifting and stoping in the Cripple Creek mines, Mr. Victor G. Hills gives the following data: Small drills are better for drifting, as well as stoping; they are not so fast, but take less power. A small compressor runs five 2-in. drills where formerly it ran only two 3⅛-in. drills. Moreover the baby drill is run by one man without a helper. A one-man, or baby drill, uses ⅞-in. steel for "starters" and ¾ in. for long drills, handling a drill 8 ft. long, although holes are usually 4 or 5 ft. deep, seldom over 6 ft. The baby drill is mounted on a bar. One driller clears the way, sets up, etc., and averages 39 ft. per 8-hr. shift in highly indurated andesite breccia (hardness, 6 to 7; sp. gr., 2.4 to 2.7). Machine men receive $4, miners $3 for 8 hrs. at Cripple Creek.

A man driving a 4 x 7-ft. cross-cut puts in 9 to 12 holes for a shift. Records by the month, without allowance for sickness, holidays, breakdowns, etc., show a progress 4.3 to 4.7 ft. per day of two shifts. A man frequently makes 3 ft. a day for several days. In making an upraise 4 x 8 ft. a man averages 2 ft. a day, doing his own temporary timbering; regular timber men following 10 to 15 ft. behind, providing a safe storing place.

In sinking a shaft 8.2 x 17.1 ft., depth, 1,000 ft., with no water to pump, two baby drills sank 5 ft. every 2 days; the machine men working only 12 hrs. in every 48 hrs., 12 hrs. being required to muck and timber; 5 muckers worked one shift to remove waste. This is an excellent record.

Confirming the foregoing data relative to the economy of small one-man drills, a paper by Mr. B. L. Thane in *Trans. Am. Inst. Min. Eng.*, 1899, may be cited. Baby drills mounted on tripods were used in stoping a quartz vein 1 to 3 ft. thick. A platform of lagging was laid on the muck to support the tripod. Holes should not be drilled more than 3 to 6 ft. deep to avoid sticking or binding. Upon striking a "slip" shorten the stroke, loosen bolts and let drill shift without stopping. A driller and a helper averaged 40 ft. of hole between 9 A. M. and 4:30 P. M., breaking twice as much rock as when using hand drills.

In drilling shallow holes it is especially desirable to reduce the lost time as much as possible. To save time in changing drill bits one authority recommends riveting the two nuts on the U-bolts. Cut a key seat in the upper end, and insert a wedge having a small lug. Put in this wedge with the tapering side toward the end of the chuck so that every blow of drill tends to tighten it. Two blows of a hammer will loosen the wedge so that the drill can be removed.

Cost of Drifting and Stoping at Rossland, B. C.—In a paper entitled "The Operation of the 'Hole Contract System' in the Center Star and War Eagle Mines, Rossland, B. C.," by Carl R. Davis, E. M., read before the *American Institute of Mining Engineers* the following data were given:

The mines in question contained a vein material that was exceedingly variable in hardness and in width of deposit, so much so that it was not practicable to measure the progress of a gang of miners by the linear foot of heading or stope. An attempt was made to keep account of the progress of each gang by counting the car loads of ore sent out, but this had to be abandoned because of the impracticability of keeping separate the ore broken by each gang, since the ore broken by several gangs of men was drawn off through the same loading chute in the mine. Finally Mr. Davis, having observed that the number of feet of hole drilled (by two miners) with a power drill did not vary 5 per cent. from month to month (although it varied greatly from day to day), decided to introduce a system of paying the miners according to the number of feet of hole drilled each day. The drillers worked in two 8-hour shifts, two men to a shift, the four miners thus forming a contract gang. Verbal contracts were made each month with each gang as to price per foot drilled. No charge was made for repairs to tools. A blasting crew of a few picked men worked from 1 A. M. to 7 A. M. while no other miners were at work, thus avoiding delays due to smoke accumulations.

Under the old wage system the miners received $3.50 a day, while under the new "hole contract system" they earn $4 to $4.25 a day, and the cost of stoping out ore is about 48 cts. per ton now, as compared with 86 cts. under the old wage system!

The following tables give details of cost:

Comparative Cost of Stoping Ore.

	System	
	Contract (49,849 tons) per ton.	Wage (13,818 tons) per ton.
Drilling	$0.356	$0.750
Blasting	.021	
Explosives	.100	.115
Total per ton........	$0.477	$0.865

Cost of Driving Headings.

	Per lin. ft.	
Drilling	$5.36	$8.36
Blasting	.68	
Explosives	2.75	2.78
Total per lin. ft	$8.79	$11.14

Four miners in drifting and cross-cutting average per month of 30 days a progress of 97.5 lin. ft. under the contract system, as against 50.8 lin. ft. under the wage system. Twelve men (3 shifts) shaft sinking averaged per month 58 lin. ft. under the contract system, as against 27.2 lin. ft. under the wage system.

It will be seen that the introduction of the contract system almost exactly doubled the efficiency of the men, whether they worked at stoping, drifting or sinking, while at the same time their daily pay was increased about 20 per cent.

The figures are eloquent enough without further comments on the advantages of the contract system both to the miners and to the mine operators.

Cost of Development and Stoping, Centre Star Mine, B. C. —The following data have been abstracted from the recent reports of Mr. Edmund B. Kirby, Gen. Mgr. Centre Star Mine, Rossland, B. C. The main shaft sinking was done in 1901; the rest of the work was done in 1903.:

Development Work.

	Main Shaft.	Small Shaft.	Raising.	Drifting.	Stoping.
Total advance, feet.......	337	79.	186.	2,903.5	......
Ore stoped, tons.........	....	....	....	......	84,453.
	Cost per ft.	Cost per ft.	Cost per ft.	Cost per ft.	Cost per ton.
1. Drilling	$12.95	$6.10	$7.31	$4.53	$0.405
2. Blasting	4.89	2.48	2.40	1.08	.03
3. Explosives	3.91	3.13	3.72	2.72	.145
4. General mine supplies..	2.35	.51	.64	.43	.04
5. Mine lighting—Candles	.62	.26	.19	.14	.015
6. " " electric	.72	.30	.22	.13	.01

7. Smithing	$1.09	$1.00	$1.14	$0.72	$0.065
8. Tramming and shoveling—direct	19.66	5.51	.65	1.21	.24
9. Tramming and shoveling—apportioned	1.54	.64	.35	.42	.085
10. Timbering—labor	9.60	1.81	3.08	.02	.19
11. " material	3.16	.33	.57	.01	.11
12. Machine drill fittings and repairs	1.15	.86	.94	.60	.055
13. General mine labor	7.81	1.57	1.18	.84	.09
14. Hoisting—underground	14.28	4.79	....	....	.....
15. " main shaft	2.19	1.48	.89	.94	.19
16. Compressed air	2.03	1.74	2.08	1.07	.12
17. Mine ventilation	2.29	.23	.17	.13	.015
18. Pumping	....	1.71	1.09	.34	.035
19. Assaying	.01	.55	.47	.14	.03
20. Surveying	.59	.20	.17	.11	.01
21. General expense	9.41	3.57	2.71	1.51	.185
Total	$99.16	$38.77	$29.97	$17.09	$2.065

The raises and drifts are evidently large in size.

The scale of wages is:

Shift boss	$5.00	Machinists	$4.00
Shaft men	4.00	Blacksmiths	4.00
Machine men	3.50	Blacksmiths' helpers	3.00
Timber men......$3.00 to	3.50	Hoisting engineers, $3.00 to	4.00
Shovellers and car men	2.50	Powder and tool boys	2.50
Carpenters	3.50	Surface laborers	2.50

The surface men work ten-hour shifts, except carpenters, who work nine. On May 1st, 1899, the Provincial Eight-Hour Law went into effect; its effect is to limit underground work to eight hours. At first, at the War Eagle Mine, this was applied by putting on three shifts in 24 hours, which reduced the actual working time of the men to 7½ hours, as half an hour was allowed for lunch. This was a wasteful arrangement, as in spite of the blowers for forcing out the smoke after blasting, such huge charges as were used to break the rock sometimes rendered a stope unapproachable for several hours. An air drill was necessarily often idle also, to permit the machine men to load, blast and then clear the muck away sufficiently to set up again.

In the spring of 1900 the three-shift system was abandoned in favor of the two-shift, arranged thus: The first shift started at 7 A. M. and worked, with an hours' intermission, till 4 P. M.; the second shift started at 4 P. M. and, with an hour for supper at 6 P. M., finished at 1 A. M. From 1 A. M. to 7 A. M. a special blasting gang loaded and fired all the ground drilled for breaking. By this method the working faces were freed from smoke before the machine men started work at 7 A. M.

Cost of Shaft Sinking at the Pioneer Mines.—At a recent meeting of the *Lake Superior Mining Institute,* Mr. Frank Drake read a paper, giving in detail the cost of lining a shaft with steel as compared with a timber lining. With steel at 2 cts. per lb., the cost of lining a three-compartment shaft with steel, was:

	Per ft. of shaft.
Steel in sets	$4.50
Labor, drilling and riveting	$1.29
Iron bearers, chairs and hangers	0.64
Wood lagging	2.12
Total	$8.55

With timber costing $14 per M, the estimated cost of lining the same shaft was:

	Per ft. of shaft.
155 ft. B. M., at $14	$2.17
Iron and bolts	.43
Labor framing sets	.32
150 ft. B. M. 2-in. lagging, at $14	2.10
Labor, cutting lagging	.27
Bearers and hangers	.28
Total	$5.57

Mr. Drake gives the cost of shaft sinking in two shafts sunk at the same time in the same material, "Ely greenstone," but, unfortunately, one shaft was sunk by day labor, while the other was sunk by contract, which makes comparison of little value. It will be seen that "B" shaft required about 23 per cent. less excavation than No. 2 shaft, due entirely to the saving in space effected by using steel.

	Shaft "B." Pioneer Mine.	Shaft No. 2 Savoy Mine.
Dip	70°	83½°
Material of sets	Steel	Wood
Material of lagging	Part wire rope and part wood	Wood
Size of Timbers of sets	....	10 x 12 ins.
Compartments	(2) 6 x 6 ft. and (1) 5 x 6 ft.	(2) 6 x 6 ft. and (1) 4 2-3 x 6 ft.
Outside shaft dimensions	6½ x 18 ft.	7 2-3 x 20 ft.
Outside area	117 sq. ft.	153 sq. ft.
Rock excavated per ft. of shaft	4 1-3 cu. yd.	5 2-3 cu. yd.
Progress per working day	1.43 ft.	1.54 ft.

Items of Cost per Foot.

	"B" shaft (steel).	No. 2 shaft (wood).
1. Contract price of excavation	$15.95	.....
2. Company labor account of excavation	5.74	$22.60
3. Labor account of lining	2.03	1.45
4. Shop and team labor	1.74	2.03
5. Explosives	1.86	1.98
Timber and lath for sets	.....	5.26
Mining timber	1.87	.63
Iron and steel	1.52	1.15
Steel rail	4.70	1.10
Wire rope for lining	.37	.....
Total, influenced by kind of lining	$35.78	$36.20
Other items not influenced by kind of lining:		
6. Miscellaneous labor regularly employed*	$10.71	$8.19
7. Pipe and fittings	1.17	.88
8. Miscellaneous supplies	4.68	2.81
9. Fuel	4.30	2.93
10. Air	2.89	1.33
11. Temporary surface buildings and head frame	.28	.58
Grand total	$59.80	$52.92

* Engineers, firemen, landers, etc.

N. B.—Fuel and air items were roughly estimated, and are subject to much uncertainty.

Here we have a comparison of cost very favorable to the steel-lined shaft, but it must constantly be borne in mind that the steel-lined shaft was sunk by contract labor and the wood-lined shaft by company labor. The "B" shaft of the Pioneer Mine, Ely, Minn., is 862 ft. deep, and dips at an angle of 70°. The explosive used in both shafts was 40 per cent. dynamite at 10½ cts. per lb. Coal cost $4 a ton. Machine men received $2.20 and helpers, $2.00 a day.

Cost of Two Three-Compartment Shafts.—I am indebted to Mr. Robert A. Kenzie for the following data on the cost of a three-compartment, vertical shaft, at the Helena-Frisco Mine, Frisco, Idaho:

COST PER FT. OF SHAFT.

Labor.		Supplies.	
Sinking	$28.69	Timbering	$4.64
Timbering	5.20	Wedges	.33
Shaft bearers	.24	Guides	.08
Hoisting	1.71	Powder	2.82
Pumping	1.44	Fuse	.66
Compressor engrs.	1.24	Caps	.06
Firing	1.41	Fuel	4.92
Station tending	1.03	Lights	.65
Tool sharpening	1.13	Tools	.56
Blacksmithing	.46	Hanging irons	.58
General expense	.55	Pump fittings	.42
		Drill fittings	.46
Total for labor	$43.10	Lubricants	.26
		Total for supplies	$16.42

This shaft was sunk from the 600-ft. level to the 1,600-ft. level, 1,000 ft., in 516 days, at a cost of $59.52 per ft. for labor, materials and supplies. The 200 ft. between the 600 ft. and the 800 ft. level, were sunk in 81 days, at a cost of $49.68 per ft. for labor, and $16.71 for supplies. Unfortunately at the time of going to press I have not received the rate of wages and prices paid.

In the *School of Mines Quarterly,* 1904, Mr. R. A. Parker gives the following data on the sinking of a shaft at the Hamilton-Ludington Mine, Iron Mountain, Mich. The shaft is 7 x 21¼ ft. in the clear; 9½ x 24 ft. being the average area excavated, and is 1,400 ft. deep in a uniform dolomite that drills and breaks readily. Two shifts were worked requiring a total of 37 men, including surface men. The best month's progress was 90 ft. The center cut holes were 9 ft. deep; the second set, 3 ft. back, 8 ft. deep and more nearly vertical; the third set, 3 ft. back of the second, 7 ft. deep, and the remaining holes squared up. The total number of holes was 32. The advance at each blast was 5 ft., 200 lbs. of 45 per cent. dynamite being used in a blast, the cut holes charged one-third heavier than the others.

Cost of Mining, Douglas Island, Alaska.—For the following data I am indebted to Mr. Robert A. Kinzie, Asst. Supt. Treadwell Mines, and to an excellent paper by him in the *Trans. Am. Inst. Min. Eng.,* 1903.

The ore is a mineralized syenite having a specific gravity of 2.61 at the Ready Bullion Mine, 2.64 at the Mexican Mine, and 2.67 at the Treadwell and 700-Foot Mines. The country rock is a slate. No timbering is used in the stopes, where the ore is shot down in large, thin slabs, so that the fall will break it as much as possible. In stoping, the drill holes average 8 ft. deep; a (3¼ to 3⅝-in.) drill averaging 28.7 ft. of hole per 10-hr. shift, breaking down 35 tons of ore with a consumption of 12½ lbs. of 40 per cent. dynamite. The cost of breaking the ore after it has been blasted is a large item of expense, as one man takes one day to sledge and bulldoze 35 tons of ore, using 0.85 lb. of 70 per cent. dynamite for bulldozing each ton of rock. A stope is usually 7 ft. high, 150 to 300 ft. long, and of variable width; the floor of a stope slopes from the parallel lines of chutes at an angle of 30°.

Drifts and cross-cuts are usually 7 x 10 ft. in the clear, and raises are 6 x 8 ft. in the clear, no timber being used. The rock has no seams or slips to break to, making it difficult to break to the bottom of holes. In drifting, 6 cut holes, 6 ft. deep, converging toward a point, are drilled; 4 relievers.

5 ft. deep, are next drilled with a slight rake toward the center; finally, 12 trimmers, 5 ft. deep, are drilled around the floor, sides, and top of the drift. The cut holes are blasted with 70 per cent. dynamite; the rest with 40 per cent. The price of the 40 per cent. powder is about 14¾ cts. per lb.

At each level a station is cut the width of the shaft, 50 ft. long, and 8 ft. high; beneath the floor of each station an ore bin is cut out so as to give a storage capacity of 500 to 1,500 tons of ore. The ore is trammed from bins at the bottom of stopes to the storage bins at the shaft, drawn off into skips and hoisted to the surface. Tramming is done by hand, by horses and by the tail rope system.

About 58 per cent. of the total ore excavated is taken from open pits, the largest of which has now reached a depth of 450 ft., and has a maximum length of 1,700 ft. and a width of 420 ft. When a pit is opened, a raise is put up from the nearest level to the surface, and all the ore excavated in the pit is shot down this raise into a chute from which it is trammed to a bin at the shaft, and hoisted in skips. The spacing of the holes in this pit work is given on page 216.

The duty of drills and of explosives in the different class of work is as follows:

	Per drill per 10-hr. shift. Ft. of drill hole	Per drill per 10-hr. shift. Tons of ore broken	Ft. of hole per ton.	Lbs. dynamite per ton.*
Open Pits	37.4	69.7	0.54	0.23
Stopes	30.0	35.7	0.83	0.36
Drifts	36.8	11.7	3.15	1.40
Raises	32.4	9.4	3.45	1.50
Shafts†	32.8	16.4	2.00	1.40

* This includes only the dynamite used in drill holes, and does not include dynamite used in bull-dozing.

† Three of the shafts are three-compartment, and one is four-compartment.

The cost operating each drill per 10-hr. shift is as follows:

Labor	$7.58
Drill sharpening, repairs, power, etc.	2.42
Total	$9.00

The cost of power is low, for about 80 per cent. of the year water power is used to run to the compressors. On the other hand wages are fairly high, being as follows per 10-hr. shift:

Machine drillers,	$2.50 to $3	with board and lodging.
" helpers,	$2.25	" " " "
Mine laborers,	$2.00	" " " "
Blacksmiths,	$4.00	" " " "
Tool-sharpeners	$3.50	" " " "

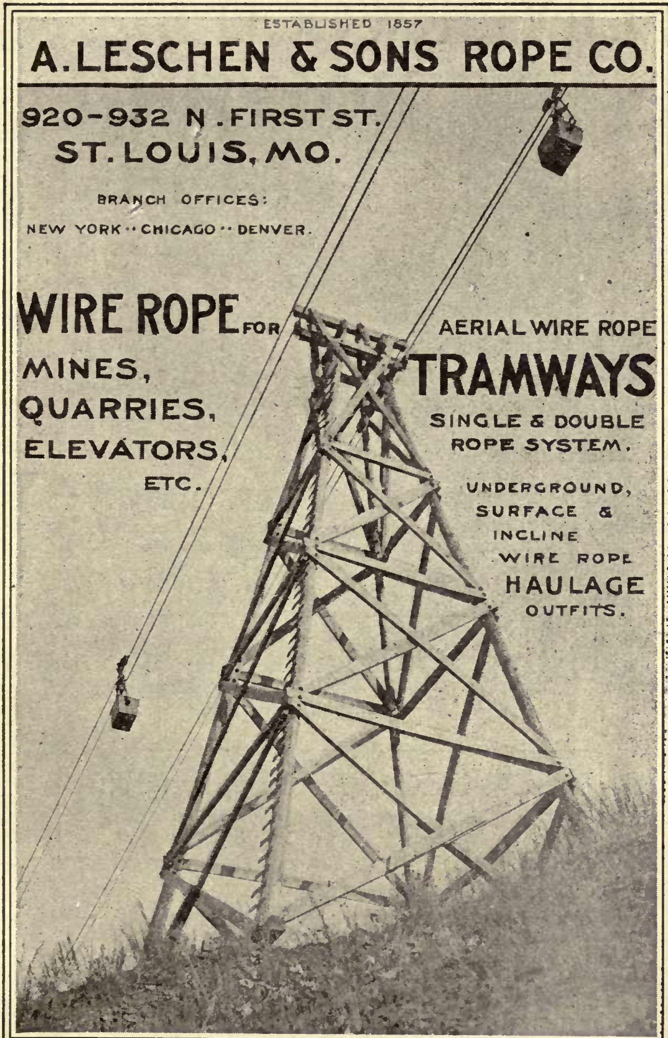
The Treadwell group of mines is justly celebrated for the fact that gold ore having a value of less than $2 a ton is mined and milled at a profit. The following table gives the itemized cost of mining a ton of ore at each of three mines, the output being that of 12 months, 1901-2. The low cost of the Treadwell mine is due not only to the fact that larger quantities of ore were handled, but to the fact that most of the pit mined ore was taken from the Treadwell. In considering the following data, it should be remembered that the ore deposit is massive, that there is no timbering, and very little pumping. The cost, low as it is, could have been still further reduced had crushers been installed capable of receiving rocks 18 x 36 ins. in size, thus reducing the cost of bulldozing.

COST OF ORE PER TON.

	Treadwell. (682,893 tons.)	Mexican. (207,455 tons.)	Ready Bullion. (226,522 tons.)
Tool sharpening, labor	$0.0191	$0.0322	$0.0400
" " supplies	.0048	.0057	.0107

Machine drilling, labor	$0.2280	$0.2720	$0.2521
" " merchandise	.0050	.0058	.0015
" " steel	.0192	.0209	.0159
Compressed air, labor	.0032	.0031	.0029
" " steam	.0293	.0572	.0882
" " supplies	.0082	.0245	.0064
Explosives, fuse	.0030	.0028	.0027
" caps	.0018	.0012	.0012
" 70% dynamite	.0079	.0175	.0154
" 40% dynamite	.0656	.0822	.0709
" labor on primers	.0042	.0056	.0063
Rock breaking, labor	.1643	.2137	.1439
" " supplies	.0037	.0037	.0041
" " fuse	.0070	.0067	.0064
" " caps	.0042	.0029	.0029
" " 70% dynamite	.1233	.0916	.0874
" " labor on powder	.0099	.0132	.0148
Tramming, labor	.0262	.0185	.0325
" horses	.0042	.0023	.....
" supplies	.0055	.0066	.0005
Hoisting, labor	.0170	0.448	.0194
" supplies	.0023	.0014	.0048
" steam power	.0293	.0572	.0882
Illuminants	.0085	.0149	.0104
Supplies, iron and foundry	.0201	.0350	.0578
Repairs	.0080	.0102	.0057
Assaying	.0015	.0027	.0024
Personal	.0039	.....	.....
General	.0017	.0029	.....
Total	$0.8399	$1.0590	$0.9954

INDEX.


ESTABLISHED 1857
A. LESCHEN & SONS ROPE CO.
920-932 N. FIRST ST.
ST. LOUIS, MO.
BRANCH OFFICES:
NEW YORK · · CHICAGO · · DENVER.
WIRE ROPE FOR
MINES,
QUARRIES,
ELEVATORS,
ETC.
AERIAL WIRE ROPE
TRAMWAYS
SINGLE & DOUBLE
ROPE SYSTEM.
UNDERGROUND,
SURFACE &
INCLINE
WIRE ROPE
HAULAGE
OUTFITS.